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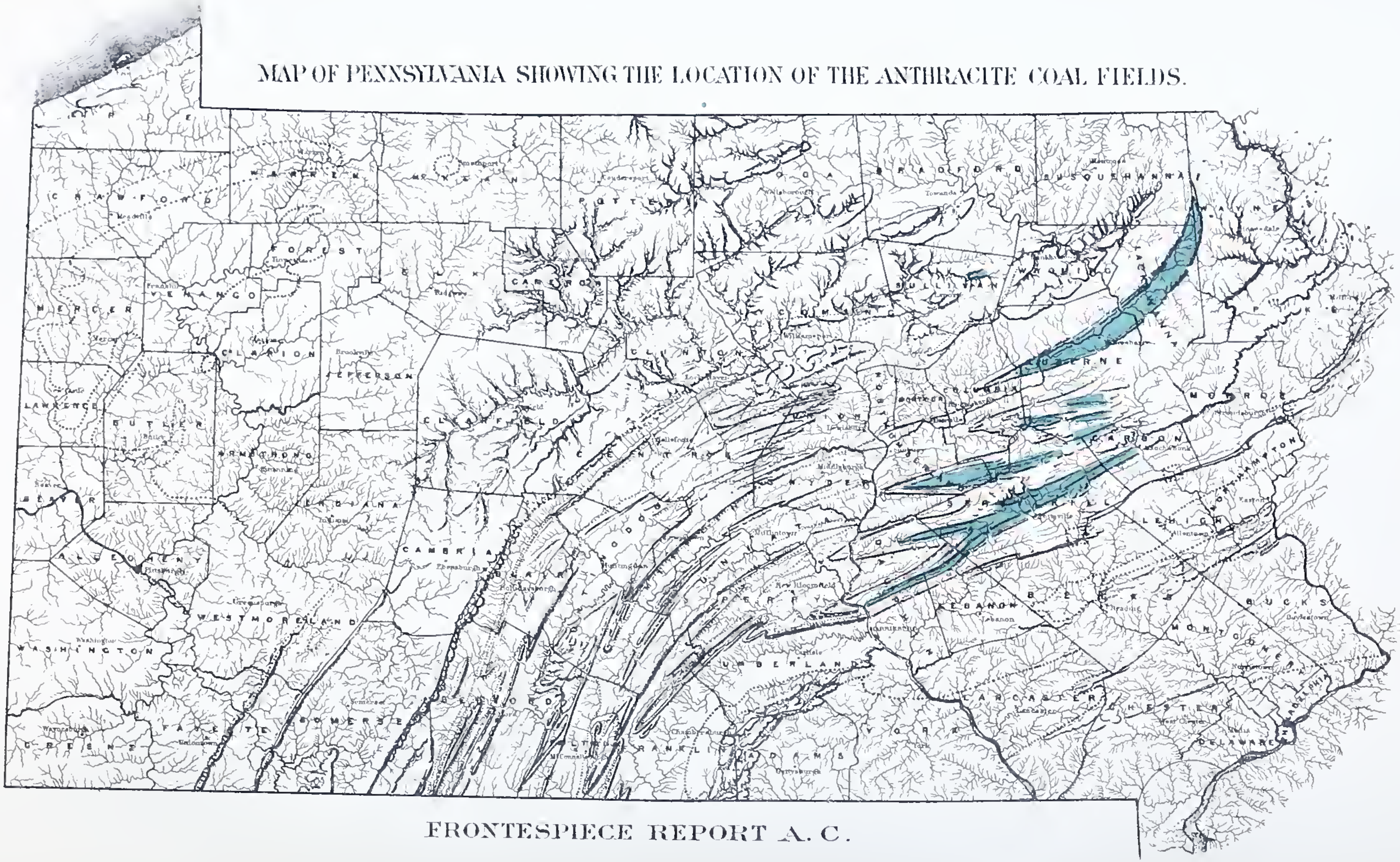

FRONTESPIECE REPORT A. C.

SECOND GEOLOGICAL SURVEY OF PENNSYLVANIA:

1883.

REPORT

ON THE

MINING METHODS AND APPLIANCES

USED IN THE

ANTHRACITE COAL FIELDS.

BY

H. M. CHANCE.

WITH AN ATLAS OF 25 PLATES;
54 PAGE PLATES;
AND 60 ILLUSTRATIONS IN THE TEXT.

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LETTER OF TRANSMITTAL.

To his Excellency, ROBERT E. PATTISON, *Governor of Pennsylvania, ex-officio Chairman of the Board of Commissioners of the Second Geological Survey of Pennsylvania:*

SIR:—I have the honor to present an important Report on Anthracite Mining Methods, prepared by Dr. Henry Martyn Chance, Assistant Geologist, who has spent more than two years in the study, description, and illustration of the subject. His ability as a geologist having been shown by his Reports on the Bituminous Coal Fields of Northern Butler and Clarion Counties, by his surveys on the Beaver river waters, and by his Report on Clinton County, he was selected for the preparation of this Report previous to and independent of the organization of the special survey of the Anthracite Coal Fields which was commenced by Mr. Charles A. Ashburner in 1881 and is still in progress.

Mr. Chance's report is intended to serve as a manual for the working of anthracite collieries in Pennsylvania, by supplying to superintendents and mining engineers such precise practical information concerning the opening of outcrops; the sinking of shafts and slopes; the construction, erection, and use of machinery; the cutting, handling, and transporting of the coal; the ventilation of the mines; and whatever else of importance is incidental to the exploration and exploitation of our anthracite beds—as the history of anthracite mining in Pennsylvania can furnish.

The board of Commissioners considered the preparation of such a book an indispensable part of the Geological Survey over the prosecution of which they presided, and the book has been prepared with all the labor and care which its great importance required. No point has been neglected. The best collieries have been selected as examples. Rival methods and machines are described and pictured. The causes and proportions of successful operations are shown by comparison. Dogmatic instruction and dictatorial advice alike have been avoided. The superintendent and engineer can see for himself what has been done or attempted at this or that colliery, and judge for himself whether others are pursuing better methods, or have better machinery than his own, or not.

Good wine needs no bush, and a book like this needs no praise—its utility will manifest itself. But I embrace this opportunity to say, that the value of this Report should be credited wholly to Dr. Chance; who comprehended its scope at the outset, devised its plan, executed it himself, designed the illustrations, edited his manuscript in press, and prepared his own lists and indexes without assistance. The book is therefore entirely his own which I now submit for your approval, and remain, sir,

Yours very respectfully,
J. P. LESLEY.

LETTER OF TRANSMITTAL.

Philadelphia, May 26, 1883

Prof. J. P. Lesley,
State Geologist:

Dear Sir:—I herewith present for your examination and approval a report on the Mining Methods and Appliances used in the Anthracite coal-fields, prepared in accordance with instructions received by me August 4, 1880.

In gathering and arranging much of the material, especially that contained in the statistical tables, I was assisted at first by Mr. Maxwell Chapman; afterwards by Mr. Arthur Winslow, who prepared some of the illustrations; and later by Mr. Otto Schwarzenbach, who executed most of the drawings in the Atlas.

The many favors and courtesies extended to us by officials, engineers, bosses and miners cannot here be individually acknowledged, but I wish to express my thanks to all who have assisted us, and especially to those who have given material aid by the loan of drawings and maps, and by giving us access to maps and records of a more or less private nature.

I have made free use of all material pertinent to the subject published in report A^2, in the reports of the Inspectors of Mines, in periodicals, and other publications.

If credit has not been given in every case the omission has not been intentional, but due to the magnitude and complexity of the subject and the exigencies of the Survey, necessitating the speedy publication of this Report before taking the field in another district.

Very Respectfully,
Your obedient servant,
H. M. CHANCE.

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REPORT

ON THE

MINING METHODS AND APPLIANCES

USED IN THE

ANTHRACITE COAL FIELDS.

CHAPTER I.

Historical Introduction.

The use of mineral coal as fuel certainly antedates the Christian era, but the date of the earliest mining operations is unknown.

A paragraph from the writings of Theophrastus, one of Aristotle's disciples, about 300 B. C., is quoted to prove its early use; but as no reference is made to mining operations, it seems probable that the coal gathered and "broken for use" was loose outcrop coal. The passage reads:

"Those substances that are called coals and are broken for use are earthy, but they kindle and burn like wooden coals. They are found in Lyguria, where there is amber, and in Elis, over the mountains towards Olympias. They are used by the smiths."

The word "coal" frequently occurring in the Bible, is doubtless used to denote wood, charcoal, or any substance used as fuel.

Coal was probably mined in Great Britain prior to the Roman invasion, but of this we have no indubitable proof; but during the period of Roman supremacy the industry must have been quite extensively prosecuted, as evidenced by the old cinder heaps still to be found among the ruins of that age.

During the Thirteenth century the English coal fields began to be systematically worked, and the coal found its way to London and other large cities, and in the early part of the Fourteenth century its use in London was sufficient to produce an objectionable amount of sulphurous smoke and soot; and finally a proclamation was issued forbidding the use of "*sea-coale*" in the city and suburbs.

Coal-mining was also prosecuted in Scotland in the Twelfth, and in Germany in the Thirteenth century; while at the antipodes the Chinese had even at that early day become familiar with the use of coal.

But it was not until the Eighteenth century that coal mining began to be scientifically prosecuted. Prior to that time the mines were of very limited depth, rarely going beneath water level; the coal was raised by a windlass or horse-gin, drainage effected by adits, or the water was raised in chain-pumps or barrels operated by hand or horse-power, and the natural ventilation—aided in some instances by falling water, and later by furnaces—was usually the sole reliance for removing foul air and explosive gases.

Yet in some of these early operations there are pictures nor unlike those to be seen every day at our modern mines; thus the following description* of the early tram-roads and wagons used at Newcastle, from "The History and Antiquities of the Town of New Castle upon Tyne," by John Brand, M. A., 1789, in which an article written by Lord Keeper Guilder, 1676, quoted below singularly resembles the present practice:

"The manner of carriage is by laying rails of timber from the colliery down to the river, exactly straight and parallel; and bulky carts are made with four rowlets, fitting these rails, whereby the carriage is so easy that one horse will draw down four or five chaldron of coals, and is an immense benefit to the coal merchants."

---

*Except Proc. Eng. Club of Philadelphia, Vol. II, No. I.

Brand also gives the details thus:

"The first thing to be done in making a wagon way is to level the ground in such a manner as to take off all sudden ascents and descents; to effect this it is sometimes necessary to cut through hills, and to raise an embankment, to carry the road through hollows. The road should be formed about twelve feet wide, and no part should have a greater descent than of one yard perpendicular in ten, of a horizontal line, nor a greater ascent than of one yard in thirty. After the road is formed pieces of timber about six feet long and six inches diameter, called sleepers, are laid across it, being eighteen or twenty-four inches distant from each other. Upon these sleepers other pieces of timber, called rails, of four or five inches square, are laid in a lateral direction four feet distant from each other for the wagon wheels to run upon; which being firmly pinned to the sleepers the road may then be filled with gravel and finished. The wagons which are in the form of a common mill hopper, have four wheels made either of solid wood or cast iron, * * * the wagon usually has a kind of trap door at the bottom which, being loosed, permits the coals to run out without any trouble. The size of wagon to carry fifty hundred weight of coals is as follows:

Length of the top	7' 9"
Breadth at the top	5'
Length at the bottom	5'
Breadth at the bottom	2' 6"
Perpendicular height	4' 3" "

The primitive brake is also thus graphically described:

"These carriages, on an easy descent, run without horses, and sometimes with that rapidity that a piece of wood, called a tiller, is obliged to be applied to one wheel, and pressed thereon by the weight of the attendant, who sits on it, to retard the motion." * * * * *

The fate of many who embarked in mining at that time is strikingly similar to that which frequently overtakes the projectors of enterprises at present, as evinced by the following, from Grey's "Chorographia" (1649.)

* * * "One merchant imployeth five hundred or a thousand in his works of coals; yet, for all of his labour, care and cost, can scarcely live by his trade; nay, many of them hath consumed and spent great estates and dyed beggars. I can remember one, of many, that raysed his estate by coale trade; many I remember that hath wasted great estates."

"Some South gentlemen have, upon great hope of benefit, come into this country to hazard their monies in coale-pits. Master Beaumont, a gentleman of great inginuity and rare parts, adventured into our mines with his thirty thousand pounds; who brought with him many rare engines, not known then in these parts—as, the art to boore with iron rodds, to try the deepnesse and thicknesse of the coale, rare engines to draw water out of the pits, wagons with one horse, to carry down coales from the pits to the stathes to the river. * * * In a few years he consumed all his money, and rode home upon his light-horse."

The first systematic coal mining operations in this country were in the Richmond, Va., bituminous coal basin. This coal was shipped to New York, Philadelphia, and Boston, in quantity, in 1790, and, prior to that date, had been in local use for smithing and domestic purposes for several years, having been used during the War for Independence, in manufacturing cannon balls, etc.

The earliest account of the finding of anthracite coal, and its recognition as such, dates back to 1768 or 1769, when it was discovered and used by two blacksmiths named Gore, in the Wyoming valley. A chronicle of this discovery may be found in a paper from the pen of Judge Fell, of Wilkes-Barre, and published in *Silliman's Journal.*

It is not strange that these smiths, having recognized the black substance as coal, should have been able to use it in their forge; but when it was tried in an ordinary open fire-place, the natural draught was not sufficient to keep up its combustion, and we consequently find that, in 1803, when about two hundred tons of anthracite were sent to Philadelphia by the "Lehigh Coal Mine Company," no one knew how to burn it, and it was pronounced worthless.

In 1812 Col. George Shoemaker shipped a few (nine) wagon loads of coal to Philadelphia from the Schuylkill region, and this was the first anthracite coal successfully burnt for manufacturing purposes without an artificial blast. The story is old and familiar: The workmen, tired of *turning* and *raking* and *fanning* the fire, closed the furnace doors and went to dinner; on their return they were amazed to find an intensely hot fire,—and the way to burn anthracite without an artificial blast was thus accidentally discovered. This occurred at the Fairmount Nail and Wire Works. Three or four years prior to this Judge Jesse Fell, of Wilkes-Barre, had succeeded in using it in an open grate, but the importance of this discovery was not at that time generally appreciated.

The *practical* mining of anthracite dates from 1820 in the Lehigh region, when 365 tons were shipped to Philadelphia,—one ton per day, in striking contrast to the present shipment of about 12,500 tons per day,—from 1829, in the Wyoming region, when 7,000 tons were shipped via Delaware and Hudson canal, then just completed; and from 1822 to 1825, in the Schuylkill region, when 6,500 tons were shipped by the Schuylkill Navigation Company.

The discovery of coal in the Lehigh region was made by Phillip Ginter while out hunting. This was in 1791. It is claimed that Ginter's discovery was not entirely accidental, as he had for some time been searching for traces of coal, having learned of its existence in the Wyoming valley.

There is a tradition given to the public in Daddow and Bannan's work on "Coal, Iron, and Oil," that the first discovery of coal in the Schuylkill region was made by a hunter named Nicho Allen, in 1790. The earliest *authentic* account is of the discovery and use of stone coal by a smith named Whetstone, in 1795. From this date until 1812, when Col. Shoemaker took his nine cart loads to Philadelphia, attempts to mine and sell the coal from this region were unsuccessfully made until the regular trade in it commenced in 1820

Geographical Introduction.

The area of the Anthracite coal-fields is relatively small; not quite five hundred square miles are included within the outcrop lines of its lowest coal bed.

The small state-map (frontispiece) and the large map (Atlas Sheet I,) show how the coal measures lie in a series of parallel basins extending through parts of Schuylkill, Carbon, Northumberland, Luzerne, and Lackawanna counties.

Geological and topographical boundaries naturally subdivide the Anthracite basins into four sub-groups (see the Geological map, Atlas Sheet No. I), thus:

1. The Southern coal-field including all of the main trough and its western fingers, extending from near the Lehigh at Mauch Chunk almost to the Susquehanna river, and embracing the Lykens Valley, Pottsville, Tamaqua, and Panther Creek districts.

2. The Western Middle coal-field.

3. The Eastern Middle coal-field embracing all that group of basins of the Hazleton district.

4. The Northern coal-field, extending from Shickshinny to Carbondale, including the Wilkes-Barre, Pittston, and Scranton districts.

Each of these basins is bounded by an elevated outcrop rim of the conglomerate forming the floor of the coal measures (No. XII), surrounded by the outcrop of the underlying Mauch Chunk red shale (No. XI), from beneath which comes up the outcrop of the Pocono Sandstone No. X, typically developed in the Second Mountain at Mauch Chunk, and in the gap south of Pottsville.

The mountains formed by the Conglomerate No. XII, surrounding the coal areas, rise to a height of from 1400 to 2000 feet above tide water, and from 700 to 1200 feet above larger streams of the region.

Over the central areas of the large basins the elevations range from 550 to 1200 feet above tide water.

The Pottsville basin has several natural outlets to the south through the gaps in Sharp Mountain made by the Little Schuylkill, the Schuylkill river, and the Swatara (Rausch creek.)

The rim of the Mahanoy Basin is broken near Ashland by Mahanoy creek and at Shamokin by Shamokin creek.

The Lehigh basins occupy a plateau, so that railroads furnishing outlets to market have a considerable descent to the low country.

The natural outlets to the Wyoming Basin are at the points of ingress and egress of the Susquehanna river through the northern rim of the basin.

The Southern Coal-field consists of the deep synclinal trough north of Sharp Mountain and a number of minor flexures lying north of this main axis. Its Eastern extremity—the Panther creek basin—may be considered to extend from near Mount Pisgah westward to the Little Schuylkill river, including a number of sharp anticlinal and synclinal folds.

Between Patterson and Middleport the field is split in two by the Mine Hill axis; on the north side of which the *Mine Hill basin* extends 15 miles to Monterey; while the *Pottsville basin* proper is widened by the addition to its southern or Sharp mountain side of two subbasins, one of which becomes the deepest part of the field at Pottsville. Between Minersville and Donaldson the Pottsville basin is contracted again to 3 miles by the loss of its two northernmost subbasins successively; and west of Donaldson forks into what is called the "fish tail"; the northern lobe of of which is the Bear Creek basin, 13 miles long and 1½ miles wide, ending at Wiconisco; the southern lobe—the Dauphin county basin—is more contracted but of greater length; the entire distance from end to end of the Pottsville basin being about 55 miles.

The *Mine Hill basin* ends in two short prongs west of Monterey; and north of it, on the plateau of the Broad mountain, are several small basins, one of which—the *Boston basin*—is half a mile wide and 10 or 12 miles long.

The Western Middle Coal-field contains six basins, arranged in two groups of three each; the southern and eastern *Mahanoy* group extends from the head of Locust valley to Ashland; the northern and western *Shamokin* group extends from the head of the Catawissa valley to Trevorton.

The two northern basins of the Shamokin group project eastward into Catawissa valley. The southern basin of the Mahanoy group projects westward into Mohontongo valley. Where the two groups lap over each other the breadth of the field is about 4 miles. Its whole length is about 40 miles.

The Eastern Middle Coal-field includes the Beaver Meadow, Dreck Creek, Hazleton, Stony run, Tomhickton, Black Creek, Macauley (McCauly ?) Little Black Creek, and Green Mountain basins ; of which the Beaver Meadow, Hazleton, and Black Creek basins are the most important. These basins are narrow, parallel, canoe-shaped troughs, complicated by minor flexures. The Beaver Meadow basin is about fifteen miles long by one half to one and a half miles wide ; the Hazleton basin is about thirteen miles long.

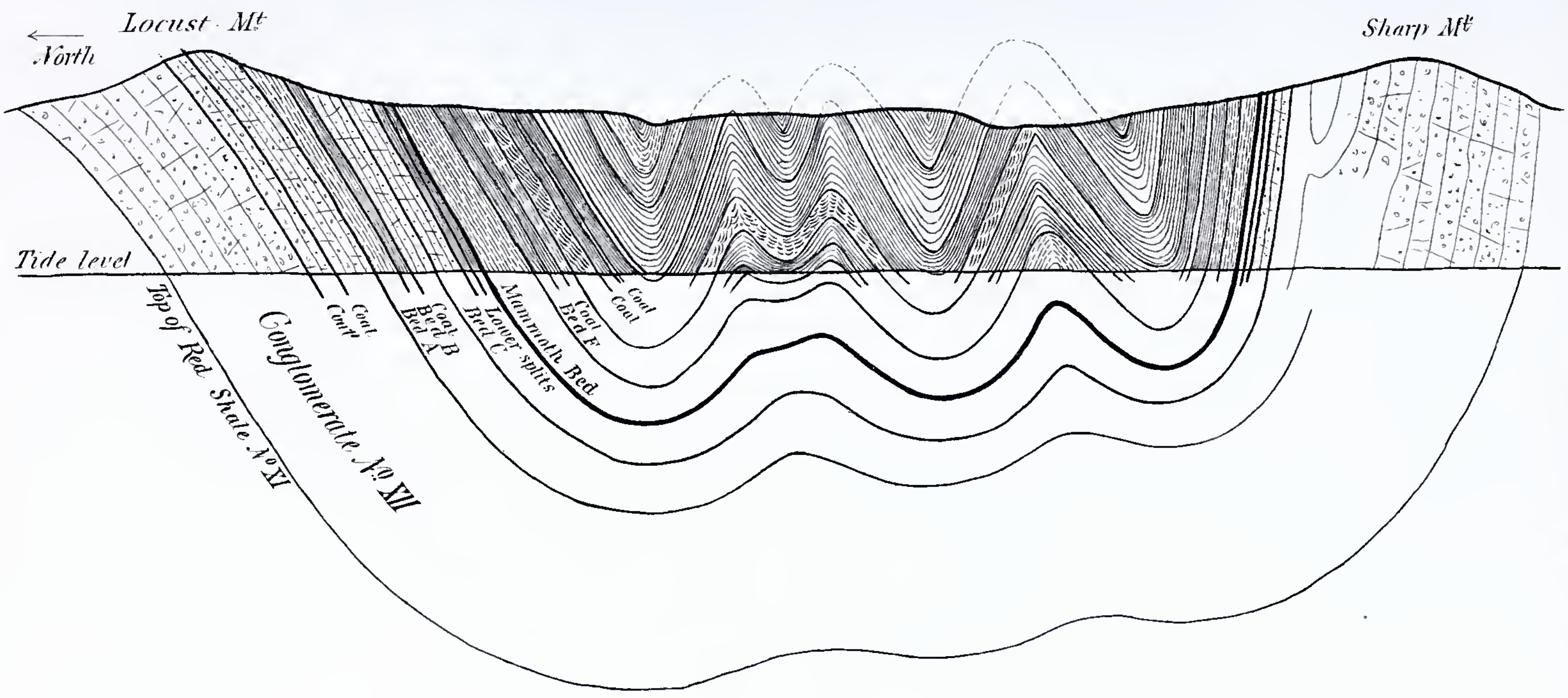
The Wyoming Basin is about fifty miles long and from two to six miles broad. Its shape is that of a crescent with the cusps pointing to the north and to the west. Its outline is regular, being free from marked indentations or projecting spurs. It is subdivided by a series of gentle anticlinal axes into a large number of sub-basins running diagonally across it east and west. Large areas of flat coal occur, and the general prevalence of low dips (rarely exceeding thirty degrees), give a distinctive characacter to the mining methods adopted in this basin, contrasting with those in general use in other fields.

The coals of this basin are now principally mined through shafts, and the mine cars are taken directly into the breasts close to the working face ; in the Second and Lehigh basins slope openings largely predominate, and the coal is generally loaded through shutes into cars standing on the gangway below.

Geological Introduction.

As a more or less comprehensive knowledge of the geological structure is necessary to appreciate the difficulties presented by any mining district, the following brief summary is necessary.

GEOLOGICAL SECTION AT TAMAQUA.



The larger anthracite basins appearing on the map as simple canoe-shaped troughs with occasional projecting fingers, are complicated by a series of minor rolls, warping the coal into curiously curved surfaces with dips ranging from the vertical to horizontal. This is is graphically shown by the cross section of the Southern or Pottsville basin (Page-plate 1), and of the Woyming Valley, (Page-plate 2.)

The subordinate axes occupying the center of these basins are often curiously disconnected; two synclinals apparently continuous with each other may be unexpectedly separated by a sharp anticlinal; an anticlinal sometimes suddenly dies away, or its prolongation is bent sharply to the north or south, and the main axis is replaced by another rising abruptly a short distance from the place of its unexpectedly sudden termination. Again, an anticlinal may suddenly become an overturned flexure, with the coal folded directly back upon itself, or the increased dip-angle may perhaps give rise to a longitudinal fracture resulting in a more or less well-marked fault.

The minor flexures usually run in a direction more nearly east and west than the average trend of the major axes, obeying in this the law governing the course of the minor anticlinal axes of the Pennsylvania Appalachians.

The number of subordinate axes bears some relation to the width of the basin, the smaller basins containing fewer minor flexures, than the large troughs.

Around the rim of each coal basin, the coal dips in towards the center of the trough, at an angle usually ranging from twenty to sixty degrees, flattening out as the central line is reached, or rolling over in a series of more or less well-defined anticlinal and synclinal waves, which, compared to the size of the basin, are mere crimples; but any one of these crimples may rise to the dignity of a well-defined axis, bisecting or bifurcating the main trough into two distinct sub-basins. Vertical and overturned dips most frequently occur along the course of these minor flexures.

Such, in brief, are the main structural peculiarities of the anthracite measures, and when contrasted with the wonderful regularity and flatness of the bituminous coal measures throughout the Appalachians, one reason why the mining of anthracite requires engineering skill far in advance of that necessary in the bituminous regions, becomes at once apparent.

The anthracite coal measures consist of an alternation of hard, massive sandstones and conglomerates, with softer black and bluish slates and shales, more or less evenly distributed. It is not possible to base any sub-division of these rocks upon the predominance of hard sandstones and conglomerates at certain horizons, for these may occur at almost any horizon ; but we may state, in general terms, that the upper part of the coal measures usually consists of softer rocks than the lower sub-divisions. The hardest rocks are, of course, found in the Conglomerate at the base of the coal measures proper.

Comparing these rocks with rocks of the same age in the bituminous coal areas, we find that they are very much harder, and more silicious, often presenting a somewhat metamorphic appearance.

This is one of the reasons why the rope drilling method of the oil regions has not more rapidly displaced the diamond drill in boring deep vertical prospecting holes.

Structural peculiarities of individual seams.

Anthracite, like bituminous coal beds, are subject to the occurrence of irregularities in the roof or floor cutting out a portion of the bed, called by the miner "*horse-backs,*" "*horses,*" or "*hog-backs*" ;* to a contraction of the seam by close approach of the roof and floor, a "*pinch*" or "*squeeze* ;" or a *bona fide* thinning away at the bed ; and to "*faults*" described by the miner as "*rock faults,*" "*slate faults,*" etc., which are not in reality true faults, as that term is understood by geologists, but a partial or entire replacement of the coal by sandstone, slate, conglomerate or fire-clay.

A fault is often called by the miner a "*thrust*" or

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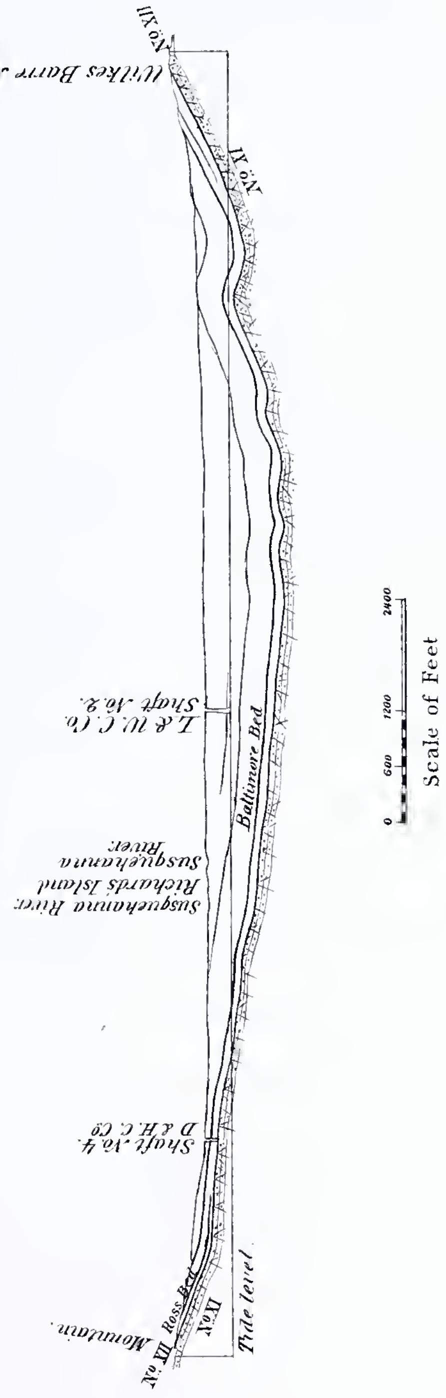
* See Glossary.

SKETCH SECTION ACROSS THE WYOMING BASIN NEAR WILKES BARRE, PA.

LOOKING EAST.



"*throw*," thus "*down-throw*," "*up-throw*," etc. They are extremely uncommon in the bituminous coal area of Pennsylvania; I know of but two; one near Brady's Bend in Armstrong county, possibly the result of a landslide—the other is described in the report on Clarion county VV.

They are not uncommon in the anthracite regions, but the amount of movement is generally quite small and the dislocation seems to be confined to the measures immediately inclosing the bed in which the fracture occurs.

These irregularities are shown better than they can be described by the illustrations now being prepared for the geological reports.

When a coal seam is cut sharply off by a fault it is of great importance to determine whether the workings are on the up-throw or down-throw side of the fracture. It may happen that the coal presents a smooth, clean cut face at the line of fracture, giving no indication whatever of the probable position of the other half of the bed, whether above or below, but the line of fracture usually displays some indication of the probable direction of movement. In the vicinity of the fault the coal may be sharply twisted downwards or upwards, or a "tailing" of fine coal mixed with broken rock may be found occupying the fissure. When these indications fail, recourse is had to geological cross sections—and these are only valuable if key-rocks can be indentified on opposite sides of the break—or to bore holes or prospecting tunnels.

Areas of crushed coal are not as common as we might *a priori* expect from the contorted condition of the coal measures. Crushed coal is commonly found along the crest and in the trough of sharp flexures, but occurs also in many other situations. The loss in mining from this cause is often very great, much of the coal going direct to the dirt dump. When the manufacture of artificial fuel or the consumption of dirt under boilers constructed for that purpose creates a *demand* for fine coal, the percentage of waste from this cause will be greatly reduced.

Dirt fault. This name is applied to an area of crushed coal, or to a partial or total replacement of the coal by a soft carbonaceous shale or slate with more or less coal running through the mass in thin stringers. One variety, characterized by greater breadth of the foreign mass at the roof, is doubtless allied to the filled stream channel, described by Mr. John F. Blandy before the American Institute of Mining Engineers.*

Slate fault. A partial and local replacement by slate is termed a slate fault; a simple thickening of a regular slate parting is also often designated by the same term—or is called a "*horse.*"

Rock-fault. When the replacing material is sandstone or conglomerate this term is used. Vertical and steeply-inclined masses of rock having nearly parallel faces are sometimes encountered. Their origin has not yet been satisfactorily explained, but in many cases they have doubtless been formed by the filling up of fissures. This term is also improperly applied to a *pinch* or *squeeze* when the abutting walls (roof and floor) are sandstone or conglomerate.

The figures obtained from more than one hundred measurements of the different coal beds taken at random, show that the slate, bony, and sulphur partings included between the top and bottom benches, constitute on an average from one fifth to one seventh of the total thickness of the coal. These partings divide the beds into benches varying from a few inches to several feet in thickness, but it is uncommon to find any single bench exceeding eight or ten feet in thickness. While the existence of these partings is usually detrimental, it sometimes happens that a thick parting dividing a large bed into two benches that may be mined as separate beds, is a great advantage, enabling the operators to recover (mine) a much larger percentage of the coal in the ground than could otherwise be obtained. When the top bench is impure and unfit for working a thick solid parting separating it from the lower benches may form an excellent roof, and greatly decrease the cost of mining the lower benches.

The irregular distribution of these slaty and bony part-

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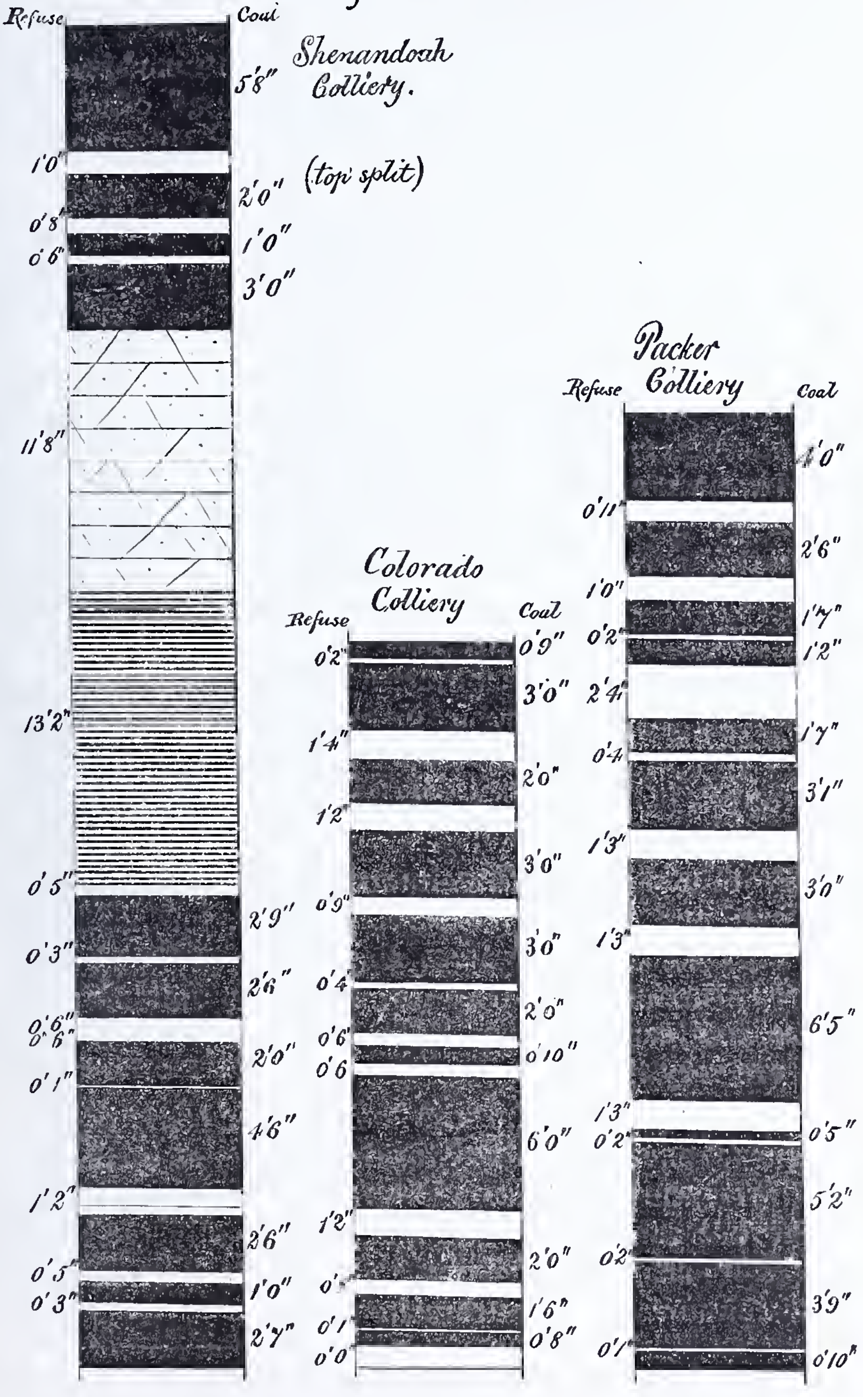
* Transactions, Vol. IV, p. 113.

Report AC, Page Plate No. 3. AC. 13

Sections of Mammoth Bed.



Scale 1" = 10'

ings is well illustrated by the sections of the Mammoth bed shown by page plate No. 3.

If we consider only the very thick beds in estimating the amount of refuse, the percentage above mentioned is apparently much too small. In the Mammoth and other large beds from one fourth to one half the total thickness frequently consists of refuse.

Thus the sections shown on Page Plate No. 3, give:

	Colorado Colliery.	*Packer colliery.*
Thickness of bed, ...	32′ 5″	42′ 5″
Thickness of coal, ...	24′ 9″	33′ 6″
Thickness of refuse, ...	7′ 8″	8′ 11″

In the forthcoming geological report illustrations will be given of seams splitting into two, three or more well-defined beds, by a simple swelling of these parting bands. That this feature is not peculiar to the anthracite is proven by similar instances recorded in the Geology of Yorkshire,* (England,) and in reports V, and VV, of the Pennsylvania survey and in other publications.

The partings found in anthracite beds are, as a rule, much harder than those occurring in bituminous seams—hence the difficulty experienced in attempting to mine anthracite by "bearing in" on a slate parting (underholing) after the fashion of bituminous mining.

Roof. The roof of anthracite coal beds is usually a dark slate, hard and tough, sometimes, however, soft and shaly, or sandstone or conglomerate. A sandstone or "rock roof" or a hard firm slate roof furnishes an excellent mining cover; but the soft carbonaceous slates and irregularly bedded shales are very treacherous.

Floor. The character of the floor is not usually of much consequence except at mines working on very steep pitching beds. It is generally hard and firm, sometimes an indurated fire-clay but sometimes slate and occasionally sandstone. The soft fire-clays occurring in the bituminous region are almost unknown in our anthracite mines.

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* Geological Survey of England and Wales, 1880.

The place of anthracite among fossil fuels.

All the fossil fuels known as coal, lignite, peat, and by other names consist of variable proportions of carbon, and volatile hydrocarbons, with a small percentage of water and a variable amount of accidental impurities.

In classifying the different varieties of coal, the impurities—water, sulphur, phosphorus, and the earthy salts constituting the ash—may be ignored, for the presence of these is accidental, and, although they greatly diminish the value of a coal when present in large amount, they do not appreciably affect the composition of the coal as fuel.

In other words, two coals containing the same relative amounts of fixed carbon and volatile hydrocarbons, should theoretically be classed together, without reference to their relative percentages of impurities.

This method of classification, devised by Prof. Walker R. Johnson, and adopted in his report on "*American Coals,*" *1844*, has recently been ably discussed in a paper by Prof. Persifor Frazer, which, with a lengthy analysis by Prof. J. P. Lesley, and annotations by Mr. A. S. McCreath, was published in the Report of Progress in the Laboratory of the Survey, (MM,) 1879.

The percentage of fixed carbon to volatile hydrocarbons may vary between the two limits 100 : 0 and 0 : 100, but neither the first, represented by graphite and the diamond, nor the second, represented by some of the purer petroleums, can be classed as coal. The coals fall between these limits.

Professor Frazer has suggested that the lines of division between coals of the different classes be drawn as follows:

Classes of Coals.	*Ratio* $\frac{C}{\text{Vol. H. C.}}$: 1
Hard-dry Anthracites,	from 99 : 1 to 12 : 1
Semi-Anthracite,	from 12 : 1 to 8 : 1
Semi-Bituminous,	from 8 : 1 to 5 : 1
Bituminous,	from 5 : 1 to 0 : 1

The commercial distinction between anthracite, semi-anthracite, semi-bituminous, and bituminous coals is largely governed by the geographical location of the mines from which the coal is obtained, and it thus often happens that a coal sold as semi-anthracite, because it is mined in or near a certain locality, really has a higher *fuel-ratio* (amount of carbon to volatile hydrocarbons) than another coal sold as hard-dry anthracite because it comes from such an anthracite district. It will be impossible to break down these arbitrary commercial divisions; but even if feasible, would it be advantageous? At present the buyer knows the coal he is using by its local or trade name; under the new classification he might think to get the same coal when ordering from the same class,—*i. e.*, coal with the same fuel ratio,—but might not be pleased with differences in physical character and in the percentage of sulphur and ash.

The following rough generalization may be useful, but its utility is limited by the fact that the coals of each class, as known to the trade, overlap into those both above and below it:

In coals known as

Anthracite,	Volatile matter is usually less than 7%.
Semi-anthracite,	Volatile matter is usually less than 10%.
Semi-bituminous,	Volatile matter is usually less than 18%.
Bituminous,	Volatile matter is usually more than 18%.

The different varieties of semi-bituminous and bituminous coals are thus grouped and described by Prof. Rogers:

Semi-bituminous	Semi-bituminous Cherry Coal.
	Semi-bituminous Splint Coal.

Bituminous	Coking Coal.
	Cherry Coal.
	Splint Coal.

Anthracite coal is characterized by its small percentage of volatile matter, high specific gravity, hardness, nearly metallic luster, rich black color, and semi-conchoidal fracture. It ignites with difficulty, produces an intensely hot fire, giving off no smoke, and burns with a very small blue flame of carbonic oxide (produced by incomplete combustion), which disappears after the coal is thoroughly ignited.

Semi-anthracite coal is neither as hard nor as dense as anthracite, its luster not so brilliant; its percentage of volatile matter is greater, and the cleavage planes or "*cleats*" are much closer, the fracture often approaching the cuboidal.

Semi-anthracite coal is confined principally to the western ends of the anthracite basins, as at Shamokin, etc.

This coal when ignited burns at first with a flame somewhat resembling that of the bituminous coals; but this shortly ceases, and the anthracitic character of the coal becomes at once apparent.

Cherry coal is a deep black, dull or lustrous coal, with a somewhat conchoidal fracture, readily breaking up into cuboidal fragments. It ignites easily, making a hot, quick fire, with a yellowish flame, and retains its shape until thoroughly consumed. Its specific gravity is much less than anthracite, about 1.30.

Splint coal is of nearly the same specific gravity as cherry coal, which it somewhat resembles. It has a slaty longitudinal and a very uncertain cross fracture. It is not very easily kindled, and makes a slow, dull fire.

Cherry and *splint coal*, of both the bituminous and semi-bituminous varieties, usually leave a rather large percentage of ash.

Cannel coal is remarkable for its dull, slaty, or resinous luster, its conchoidal or slaty fracture, and its large percentage of volatile hydrocarbons. It burns with a long flame, giving off a large amount of smoke, and usually leaves a very large percentage of ashes.

Caking or *coking coal*, the most important of the bituminous coals, is readily known by its behavior in the fire. It burns with a long yellow flame, giving off more or less smoke, and creates an intense heat when properly attended. It is usually quite soft, and does not bear handling well. In the fire it swells, fuses, and finally runs together in large masses, which are rendered more or less porous by the evolution of the contained gaseous hydrocarbons. The *cleat* and cleavage joints (butt and face) usually run nearly at right angles, giving the coal a cubical fracture.

Hard Dry Anthracite.

Coals of this class burn without smoke or colored flame (but with a faint blue flame until thoroughly kindled); they have a small, almost colorless, and intensely hot flame when thoroughly ignited; but ignition is always slow and difficult.

In specific gravity they range usually from 1.45 to 1.75; they are very much harder than all other varieties of coal, and usually contain fewer cleavage planes.

In composition they may be considered to range:

In Carbon, from	91% to 98 %.
“ Hydrogen, “	0 to 3 %.
“ Water, “	½% to 2½%.
“ Ashes,* “	1% to ?
“ Oxygen, Nitrogen, and other gases, from	0 to 2 % or 3%.

Semi-Anthracite—Soft Anthracite.

Coals of this class are softer, and are less disposed to show the conchoidal fracture of hard, dry anthracites. When first ignited they give a yellow flame resembling the flame of a true bituminous coal, but of much smaller size; but when this volatile matter is driven off and the coal fully ignited, the fire is similar in every way to that produced by a hard-dry anthracite.

Coals of this class predominate in the western parts of the Southern and Western Middle coal-fields and in the Wyoming district. They range in composition thus:

Carbon,	85 % to 90%.
Hydrogen, (principally in combination,)	
Water,	1½% to 3%.
Ashes, (including Pyrites,)	3 % to ?
Oxygen, hydro-carbons, nitrogen, etc.,	5 % to 10%.

Gases.

No examination has as yet been made by the Survey of the gases contained in and given off by anthracite coal; it is known, however, that the bulk of such gases generally consists of carbureted hydrogen (CH_4), but in some cases carbonic acid gas (CO_2) is given off in considerable quantities, and it is not unlikely that a thorough investigation might disclose the existence of hydrogen, nitrogen, and some hydrocarbons belonging to the same series with CH_4. The occurrence and discharge of gases from coal will receive attention in another chapter.

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* Including Bi-Sulphide of Iron—(Iron Pyrites.)

CHAPTER II.

Prospecting for Coal.

The term "prospecting," meaning, literally, a determination of the prospects of obtaining valuable mineral in paying quantities, may properly be considered to include all the methods employed for that purpose ; not mere surface examinations, but bore-holes, shafts, tunnels, and drifts—if these are used for exploration, and not for the extraction of mineral or coal—the methods of utilizing the information thus obtained ; the selection of samples for analysis, and the methods of estimating the amount of available or workable mineral or coal.

In the oil regions, prospecting for and "mining" oil are nearly identical operations, the one merging into the other when the operator is successful in obtaining oil in paying quantities.

Prospecting operations may be conveniently considered under the following heads :

1. Surface Examinations.
2. Prospecting Drifts, Shafts, and Tunnels.
3. Bore-holes.
4. The Construction of Geological Maps and Cross-sections.
5. Selection of Samples for Analysis.
6. Estimates of Available Tonnage.

1. Surface Examinations.

When the presence of coal is suspected in an unexplored tract of land, a thorough examination of the surface is the first procedure. This may at once result in the actual discovery of coal, in conclusive proof of its absence, or may yield only negative results.

Before commencing this examination, it is advisable to procure any maps that may be in existence of the tract to be examined, and also maps of the adjoining lands. Although these may be imperfect in detail and possess no high degree of accuracy, they will be serviceable in showing the location of streams, the ravines or valleys in which they flow, the watersheds, and mayhap other important features.

The roads are first scanned for traces of smut, and all outcrops, of whatever description, carefully noted; when the exposures are sufficiently good, the rate of dip is taken with a clinometer and its direction jotted down upon the map. Outcrops exposed along the water courses and in the stream-beds, will often contribute largely to the information already obtained.

A pocket compass and a clinometer are almost indispensable adjuncts to the successful prosecution of these researches, but if the latter cannot be obtained, it is possible to measure the dip quite accurately with a spirit-level and a foot-rule, the inclination being expressed by the ratio between the perpendicular and horizontal, as mitres are usually measured by carpenters and masons.

These observations will probably furnish data showing the existence or absence of any marked anticlinal or synclinal flexure, and may indicate the approximate location and extent of important flexures or faults. If the country is drift covered, and few exposures are visible, the streams and ravines are thoroughly and minutely searched for fragments of coal, and if any be found, the search is then to be prosecuted up stream to determine whether there be a point beyond which no fragments can be detected. The discovery of such a point is presumptive evidence that the outcrop is near. Small excavations with pick and shovel are then made along the stream bank to find and expose the coal to view. The smut of a bed having been thus discovered, a small drift is driven in a sufficient distance to fully expose its thickness and to determine its quality, and the character of its roof and floor.

When ravines and valleys are so filled with debris, and the banks so thickly covered with detritus from the hills above that no exposures are visible and nothing is gained from an examination of them, it becomes necessary to rely upon topographical indications.

The topographical features which denote the presence of a bituminous coal seam are easily recognized by anyone familiar with the peculiarities of coal-measure topography. The most prominent of these is the bench or terrace which almost invariably occurs at the outcrop.

The soft coal seams encased in harder rocks are easily and rapidly eroded, and produce, by their rapid disintegration, a series of benches following their lines of outcrop.

But as every hard stratum will also produce a terrace of some kind, it is necessary to have some means of distinguishing a coal terrace from a bench marking the outcrop of some other stratum. In the bituminous coal regions, where the seams lie in the hills with very slight—almost imperceptible—dips, the site of a coal bed is nearly always indicated by springs loaded with iron, which is deposited in ochery films upon the stones and vegetable matter over which the water flows. In the anthracite regions the beds, being highly inclined, rarely furnish such an indication of their presence, except in the sharply cut gaps and ravines eroded across the hills in which the coal occurs.

The anthracite coal terrace is often a well-marked topographical feature, but in many localities the site of the outcrop is not marked by any distinct bench or terrace, and surface examinations fail to disclose any important features. In tracing a coal terrace the breadth is always affected by:

1. The thickness of the seam.
2. The dip of the bed.
3. The slope of the ground.

When the bed dips *into* the hill, the terrace is broader than when the pitch is in an opposite direction, and when the surface slope is gentle the terrace is generally broader than where a steep contour prevails.

A good conception of the direction and strength of dip may be obtained by tracing a terrace for some distance and carefully noting its deflections from a straight line, and the relations of these to the contour of the ground. If the variation occasioned by a depression is towards the foot of the hill, the coal dips in the same direction with the slope of the ground, but if it runs in towards the top of the hill, the reverse is true.

Having found a terrace which presents the appearance of a coal bench, search is made on, and for a short distance below, the flat for some positive indication of coal. This may best be prosecuted with a pick and shovel.

Coal fragments are sometimes found in the soil, adhering to the roots of overturned trees, or in the earth thrown up at a ground-hog's burrow.

When the presence of coal is proven, or if the search be thus far unsuccessful, a systematic series of trenches may be commenced.

Where the dip of the measures is flat, the exploring is generally begun some distance *below* the supposed place of the bed, by digging a trench about two feet wide and from five to ten feet long, ranging up and down the hillside. This should be sunk three to six feet deep, and if no trace of coal be then found, the trench is usually abandoned, for either the drift (soil) is very deep or the trench is too low down on the hillside. In the former case the trench may be widened into a shaft, and sunk until some trace of coal or the solid rock is reached. If these procedures result negatively, it only remains to repeat the experiment at a higher or lower level on the hillside, and at some distance longitudinally from the first digging. The mere fact of finding no trace of coal in the loose drift is not conclusive proof that coal is not present, for the soil in which the digging is located may have been brought down from the hill above by a land-slip, sweeping before it all traces of the bed. For this reason, subsequent excavations are best located some distance from the first trial pit.

When at last small fragments of coal, or a well-defined smut or blossom is discovered, it only remains to extend the trench up hill, never losing sight of the blossom until it is traced to the bed from which it comes.

Ditching tools are sometimes used in this method of prospecting, but only a pick and spade or shovel are necessary.

When the coal measures are dipping sharply either with or away from the slope of the hillside, the process is much the same.

In flat-pitching measures it is often advisable to start a shaft a few feet above the supposed outcrop of the bed and sink down to and through the seam.

When the thickness and quality of the bed are tolerably well known and the position of its outcrop the only information desired, this laborious process may be replaced by holes bored with an earth auger. This instrument is so nearly like the coal-boring augers or hand-drills illustrated and described with the miners' tools, that a separate drawing is unnecessary. Its cutting edge has a larger flange and the screw pitch is best made somewhat steeper, but in other respects it does not differ from that tool. It is ordinarily operated by a handle keyed or screwed to the shaft which may be lengthened by attaching additional rods or segments of gas-pipe so that in favorable ground the borings may be continued to a considerable depth, but in friable soil much difficulty is frequently experienced from the caving in of loose material. This may be overcome by using augers of different diameters and lining the hole with an iron pipe, the size diminishing in regular order as greater depth is reached.

If the surface examinations yield no evidence of the presence of coal, but, by reason of the ground being occupied by rocks of coal-bearing age, the existence of coal yet appears probable, boring is usually resorted to ; but it is seldom necessary to bore holes to prove the mere existence of coal, for, if the surface examinations are carefully made, unmistakable evidences of its presence will generally be discovered.

The creep of the blossom down hill, when *away from the* *bed*, will seldom cause the crop to present an exaggerated thickness in a prospecting trench or shaft; but when the bed *dips with the hill slope*, the crop is usually overturned down hill, and the blossom is thus doubled over upon the outcrop. A shaft sunk through such an overturned outcrop would deceptively indicate the presence of a bed much thicker than the actual measurement of the seam. The shaft should, therefore, be sunk through the entire thickness of coal until the bottom clay or slate is reached, and then widened out horizontally, or a level driven at right angles to it until the top slate is encountered.

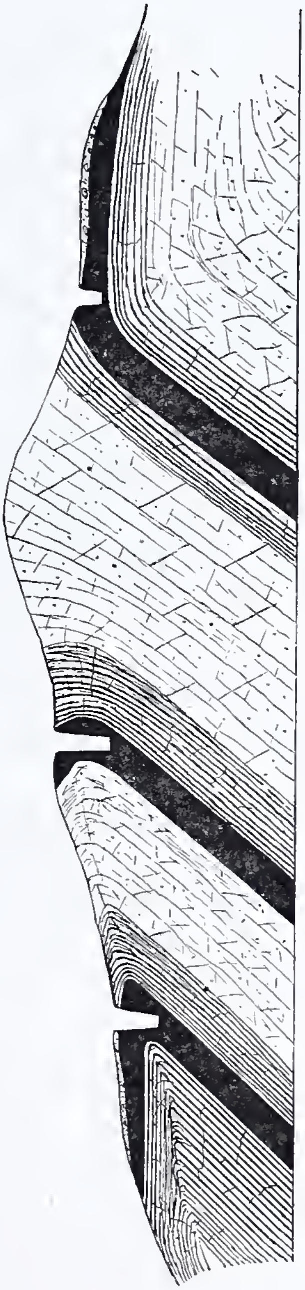
The accompanying illustration (Page plate 4) is taken from Vol. II of the final report of the First Survey. It shows the long "*tailing out*" of an outcrop down hill, and the overturning often seen when the dip is with the hill slope. This folding back of the outcrop, both of coal beds and other rocks, is familiar to most prospectors and practical miners. It is probably resultant from the combined action of frost, water, and the tendency of loose material lying on a slope to slide down hill. This downward tendency sometimes produces land-slides that carry the tailing of a coal seam several hundred feet below the outcrop of the bed.

When two sets of cleavage planes cross the coal at nearly equal angles, or when the outcrop is twisted and contorted by a creep, it may be difficult to distinguish the roof from the floor of the seam, on account of the small area of the bed exposed in a prospecting opening, and the direction of the dip remains uncertain.

The occurrence of *stigmariæ* in one of the rock walls of the seam is presumptive evidence that the stratum containing them is the floor of the seam, but if *sigillariæ*, *fern leaves*, *etc.*, are found, the rock is probably the roof or top rock, although both of these fossil plant remains *may* occur in either the hanging or foot wall of a coal seam.

When the seam has a strong dip and outcrops on a steep hillside, the prospecting shaft may advantageously be replaced by a tunnel. Under such circumstances the latter is undoubtedly superior to a shaft, and even when the con-

Section showing Prospecting Shafts on Overturned Out-crops.



Reproduced from the Report of the First Survey.

ditions are not so favorable for tunnelling as for shafting, the former method, if at all practicable, will probably be found more satisfactory in the end.

In a tunnel the water gives little trouble, the rock or débris is easily taken out in a small barrow wheeled on a line of single planks or boards laid for that purpose, and the workmen are saved the time and labor lost at a shaft in raising and lowering each other. Then again, a tunnel properly driven remains free from water, and the vein can be examined at any time in the future. But a tunnel often requires much heavier timbering than a shaft, (and a greater distance must be driven to reach hard coal,) and this is a serious drawback.

By slightly modifying the details of these surface prospecting methods, they may be applied generally in the examination of any stratified mineral deposits, and they are not altogether unsuited. to the exploration of metalliferous deposits occurring in veins or fissures and in metamorphic or igneous rocks.

Prospecting drifts, shafts, and tunnels.

A drift, sometimes from its small size called a "monkey drift," can only be economically driven in upon the coal when the bed lies above water level with a very slight dip or when the dip is sharp but the outcrop is found in a ravine or valley eroded across the strike of the measures.

When a drift is to be driven directly upon the bed, the steepest part of the hillside on which the crop can be located is selected, as at this place hard undisturbed coal will be more easily reached than where the slope is gentle and the soil correspondingly deep. When the coal lies flat the drift is commenced as a trench two feet wide and a few feet below the crop and carried uphill, constantly deepening until the base of the blossom is reached. The trench may then be widened out to the full width of the proposed entry, which in ordinary ground cannot be economically driven with a width less than four or five feet at the base. Its height had best be made at least three feet and a half, and the cost will be no greater if it is driven somewhat higher.

A height of four feet inside the timbers, corresponding to a height of about five feet untimbered, with a width at the base of five to five and a half feet (outside the timbers) are good working dimensions.

It is not advisable to attempt getting under cover too quickly, and until at least three to five feet of cover is obtained, the entry is best continued as an open cut. The floor of the drift should be given sufficient slope to insure good drainage, and it is for this purpose that the opening is commenced below the supposed base of the coal, for if the bed dips *into the hill*, a deep trench may be needed. Nothing is lost by this precaution, for even if the drift is found to be several feet too low, it is easily driven up to the coal, and the slope so obtained greatly assists the workman in wheeling out the loose material. An inclination of one foot in fifteen or twenty is advisable.

A drainage ditch 4 to 8 inches deep, and 8 or 12 inches wide, is dug on one side of the drift, or is located in the middle and covered with the boards or planks on which the barrows are wheeled.

When the seam is dipping strongly, and the drift is to be driven upon an outcrop laid bare in a ravine or cross-cut valley, there is no necessity for digging the preliminary trench or for commencing the entry below the seam. The drift is located immediately upon the bed, and driven in on a gentle grade until firm coal is found; it is then widened out to the full size of the bed, or a cross heading is driven at right angles to the drift until both the roof and floor of the bed are exposed.

It is generally necessary to use some timbering at the mouth of a prospecting drift to keep up the disintegrated rock and soil overlying the crop. The policy of building this timbering strongly and substantially cannot be too strongly urged. It costs little more to put in good, sound legs and collars, with carefully cut joints, than to do the work in a shiftless manner. If properly timbered such an entry will remain open for years, and the coal can be examined at any time; but if the timbers are light, far apart, and poorly set, they will probably be crushed in a short time—frequently in a few months.

For an entry four feet high and five feet wide at the base, the legs and collars may be made of round stuff not less than six inches in diameter, and if the entry is broader, the collar should be at least eight inches. The legs should be set in a step two or three inches deep, but the bottom of this step should, if possible, be above water-level in the ditch; when the roof is very loose and bad, they should not be more than three feet apart. Waste slabs may be used for lagging, a double layer being driven over the collars, which should be firmly wedged down upon the legs. Round stuff three or four inches in diameter makes a better lagging than slabs; but when the latter are at hand they will answer the purpose, and are cheaper.

In loose stuff the lagging is best cut long enough to lap past two legs, and each set of lagging-slabs should overlap the one next it. When the top is very loose, and at the same time very heavy, the top lagging may be cut shorter and made of heavier stuff, care being taken that the sets overlap.

In flat pitching seams of moderate thickness, it is sometimes advisable to open the drift at once, to the full height of the bed, but if the seam be very thick, this will be impracticable. Its full thickness may then be exposed to view by carrying the drift up in a series of steps, until the top rock is reached, and opening bank.

The tools needed in drift driving are one or two earth-picks and shovels, two or three coal picks, a small wheel-barrow, an ax, a single cross-cut saw, and a wooden maul for driving the wedges and lagging. If the drift is driven in so far that the air becomes bad, a small blowing fan and a set of board pipes may be necessary.

The air may be conveyed from the fan into the drift or tunnel through wooden box-pipes, eight or twelve inches square, made of inch boards, or through a rude airway made by inclining short slabs against the side of the drift and covering them with a layer of clay or earth.

When the prospecting entry is a tunnel driven across the outcropping rocks to the coal bed, the method of driving is precisely the same as that already described, but, as the entry cuts across other rocks standing at a high angle, hard strata, necessitating drilling and blasting, are to be expected. The quarryman's tools must then be used. Rock drills, the ordinary coal drill tempered for slate, needle and scraper, sledges, powder, fuse, etc., will constitute the outfit.

When drilling is necessary, two or three men will, of course, make much better time than one. This cannot be said of drifting, for in a drift one man will often do nearly as much work as two. For small drifts or tunnels in soft ground, one man can do handsomely working alone, but in hard ground, where the pick work is very laborious, it is true economy to employ two or three men.

Shafting is generally a slower method, and always more laborious and expensive than tunnelling, except when the depth to be sunk is not more than fifteen or twenty feet. The tools required are about the same as those used in drifting or tunnelling, but in shallow holes through earth or decomposed rock only picks and shovels are needed. Good laborers will sink a shaft ten or twelve feet deep without a staging, *i. e.*, a platform to throw the earth on.

When the depth exceeds twelve feet, it is necessary to build a staging or cut a step to throw the earth upon, and from which it is again shovelled, and thrown from the pit; or to erect a windlass for hoisting.

When the depth exceeds fifteen or eighteen feet a windlass becomes necessary. This may be a very primitive affair, but should be *strongly built.* A hemp rope one inch thick and a strong iron-bound wooden bucket holding one bushel or more complete the outfit.

The uprights upon which the rope-shaft rests should be securely fitted to a horizontal foot-piece and braced by diagonal braces both at the sides and back.

Some men prefer sinking square holes, others prefer round shafts. If the ground is firm the shape is of little consequence. A rectangular hole about three and a half by five feet gives sufficient working room and is for some reasons preferable to any other form.

A square shaft is best adapted to sinking in loose treacherous earth which threatens to cave in and needs heavy timbering. Theoretically a hexagonal or octagonal hole is better under these conditions, but it is difficult to get workmen to cut the joints and fit the timbering with sufficient accuracy to yield the requisite resistance to side thrust, whereas square timbering is easily fitted and not so apt to become misplaced, and is moreover much cheaper.

The timbering is kept in place either by supporting it beneath or hanging it from above, being of course in either case wedged as tightly as possible in place by wedges and by a series of boards, waste slabs or small round stuff driven in behind the timbers.

Round timber six or eight inches thick may be used for the crib-work, but square stuff is better as it can be more easily and accurately fitted.

The distance between the cribbing girts will depend entirely upon the character of the ground and depth to which timbering is necessary. It is never advisable to place them more than six feet apart and generally they should be closer together.

At least two men are always required in shaft-sinking and the work will generally progress much more expeditiously if three men are employed. It is hardly safe to allow one man to hoist or lower his fellow workman from a pit of any considerable depth on the windlass ordinarily used for this kind of work; again, as the depth increases the man hoisting earth will raise it so slowly that the laborer below will remain idle much of the time. If it be impossible to place more than two men at work on the sinking, these two will be enabled to do tolerably well, if, for the windlass generally used, a differential windlass be substituted. This consists of a shaft similar to the ordinary windlass except that one half of it is of greater diameter than the other half. One end of the rope is fastened to the larger part of the shaft, and a number of turns taken, the other end is passed down the shaft and through a single pulley block hooked to the bucket, brought up and coiled in the opposite direction around the small part of the shaft.

One end, side, or corner of the pit is kept in advance of the average level of the shaft floor, thus providing a sump or reservoir for the water. This may be hoisted in the bucket or may be raised by an ordinary suction pump when the depth is small or by a bucket-lift pump when the depth exceeds twenty-seven feet. It is not advisable to use a force pump in the shaft, as this occupies space needed for the workmen in the pit and is no better than the bucket-lift pump which is worked by the laborers on top without hindrance of any kind to the sinkers.

If the natural ventilation is not sufficient to give pure air it may be materially increased by hedging off one corner of the shaft by a broad board, and within or immediately beneath the triangular space thus formed (which should run from within two or three feet of the bottom to the top) placing a good strong lamp; but if the sides of the shaft are very wet this will not give good results. Under such circumstances the heated column of air may be turned into a board pipe twelve or fourteen inches square. A shaft forty or fifty feet deep may be amply ventilated in this way except in very warm weather.

Underground tunnelling and shafting to explore the rocks adjacent to gangways in a mine being worked, are now generally replaced by bore-holes drilled with the Diamond and other drills; but for surface explorations of moderate depth the old-fashioned methods of drifting, tunnelling, and shafting are unequalled, both in cheapness and efficiency.

Prospecting holes bored in soft ground with an earth-auger are often valuable as auxiliary to other examinations, but it is not possible by this means to determine *exactly* either the dip, quality, or thickness of a coal bed, but when the *position* of a coal outcrop is the only information desired the earth-auger is a valuable aid.

It may not be out of place here to mention the necessity of examining carefully the pitch of a bed in making measurements of thickness in a drift or entry driven in the bed.

Inexperienced persons are occasionally misled by unscrupulous speculators and others in the following manner:

A drift is driven in on a bed of coal or ore dipping say fifteen or twenty degrees, and four feet thick. The drift is driven so that *the top* on one side just touches the roof of the bed, and on the other side it nearly coincides with the line of junction between the bed and the rock underlying the coal or ore. On the side touching the roof the bottom of the entry is cut down to the floor. To do this it will at once be seen that a large amount of the bottom rock (as much rock as ore or coal) must be taken out from the other side of the drift. In addition to this a large amount of the floor of the drift,—to a depth of one to three feet, is taken out on the side nearest the bed and the cavity thus made is allowed to fill with water, while only a narrow strip of rock is left dry to walk upon on the opposite side of the entry.

Coal or ore is therefore in sight at every point except the strip of rock left to walk on and the side adjoining, and the purchaser is informed that *ore or coal*, as the case may be, was gouged out from the part covered with water. Accepting this without suspicion, he naturally concludes that the base of the bed being at his feet at the water line, and being also in sight at his shoulder, the dip is sixty or seventy degrees, and that the thickness of the bed is nearly equal to the width of the entry (eight or ten feet.)

Prospecting by boring.

When the rocks which are supposed to contain coal or other valuable mineral lie deeply buried from sight beneath a mass of other rocks the cheapest means of obtaining a knowledge of their contents is by bore-holes. Boring should never precede a careful surface examination; for even if this fails to detect any mineral deposit, it will almost invariably furnish some data which will greatly assist in locating the bore-holes where they will be most serviceable. In fact, bore-holes are, in a majority of cases, used not to detect or prove the existence of mineral or coal beds, but to define their position, to determine their thickness, or to fill up a gap in some doubtful locality, in which there are no trustworthy exposures or in which the beds lie at considerable depth beneath the surface.

The methods of sinking prospecting holes now in general use are three; two of which are true drilling processes; the third is by boring:

1. Spring-pole or oil well drilling.
2. Steam percussion drills.
3. Diamond boring machines.

For hard rock the Diamond drill certainly has many advantages over the spring-pole and oil well methods. The percussion drills are not adapted to boring deep holes.

Spring-pole drilling.

The tools used for drilling holes of moderate depth are three or four bits or drills, a reamer, a set of rods either of iron or wood, levers, wrenches, a sand-pump, and windlass.

The bits are much like, or may be made precisely like, those used for oil well drilling figured on page plate No. 5, but of much smaller size. They are made of Norway iron, tipped with shear steel or are made of steel throughout. The rods are from one half to one inch in diameter if of iron (or 1″ square or less) and jointed together by pins and sockets like those shown on the larger tools on page plate No. 5. If the rods are of wood they may be made like the sucker-rods used in oil wells, one and a half to one and three fourth inches in diameter of hexagonal or octagonal cross-section, and fitted at each end with riveted iron straps by which they are connected together by pin and socket in the same manner as the iron rods. They are made in joints of eight, twelve, or sixteen feet, with one or two shorter pieces.

A small pair of jars is sometimes used in deep holes but is much more serviceable when a rope is substituted for the rods and the tools made somewhat heavier.

A sand-pump of ordinary construction with a bucket and standing valve is used to extract the sediment.

A rude derrick or shears is built above the hole and to this the windlass is rigged.

When the surface drift or soil is of moderate depth it is customary to sink a small shaft down to the solid rock. A wooden conductor box 4″ or 6″ square is then inserted and firmly held in place by being stepped down three or four inches or more into the bed rock. If solid rock of some sort cannot be reached within fifteen or twenty feet it is customary to drive iron pipe down through the loose material to the bed rock.

The drilling is commenced in this loose stuff and continued down for some distance before the first joint of pipe is inserted, and the hole is always kept in advance of the bottom edge of the pipe. A large wooden maul or hammer held between board guides is used to drive the pipe in very much the same way that pipe is driven on such a large scale in sinking wells in the Pennsylvania oil fields.*

Although the size of the drill-hole is not a matter of very much importance to the prospector, there are certain limits within which it is always well to keep. When boring by the spring-pole system it will generally be found best to drill holes two and a half or three inches in diameter, but holes no larger than two inches and as large as three and a half or four inches are sometimes successfully drilled; but in these latter the progress is slow.

The spring pole from which the rods and drill are suspended lifts them by its rebound after every stroke. The up and down motion is communicated by hand-power or a stirrup is attached to enable the driller to use his feet; hence the expression "Kicking down" a hole.

No rules can be laid down for the speed at which the drill should be revolved to insure a round hole, or the intervals at which to withdraw the rods and sand-pump the sediments, as these will vary indefinitely with the character of the rock.

When the rods are replaced by a rope the whole operation becomes similar to the method now used in oil well sinking, but the tools are of course correspondingly lighter.

As this oil well method is now being used at some deep prospecting holes in the Anthracite regions and as it is frequently employed in other districts the following description condensed from the report of Mr. J. F. Carll* is in place here. When a hole three hundred feet deep or deeper is required, the spring-pole method will be found slow and unsatisfactory and this oil-well method of rope-drilling, modified to suit the existing circumstances, will be cheaper and more efficient.

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* I.I.I. Second Geological Survey of Pennsylvania.

Shallow holes are often drilled without a spring-pole, the drill and rods being raised by hand. A top-piece resembling a capstan head, with four handles or arms placed at right angles, is attached to the upper end of the rods, and the rods are lifted, and allowed to fall, by two men standing opposite each other. The drill is rotated by the men who walk slowly around, taking one short step at each stroke of the drill.

Sand-pumpings.

The sand-pumpings from all bore-holes should be carefully washed, dried, and preserved in labelled vials. They may prove of no immediate use, but instances are not wanting where much time and money might have been saved had they been properly preserved ; and this applies as well to diamond drill cores as to sand-pumpings.

Rope Drilling.

The following description of the method of drilling for oil is condensed from Mr. Carll's minute description in report I.I.I. Atlas Plate No. XXV, shows the details of the drilling "rig" consisting of a derrick fitted with bull-wheel and crown-pulley (sheave) for raising and lowering the tools ; heavy mud-sills with main-sill and sub-sills bearing the samson-post and jack-posts ; a walking-beam, band-wheel, and sand-pump reel ; and foundation timbers and house (not shown) for the engine.

The "rig" costs from $400 to $700 in the oil country. Its cost in the anthracite regions will approach the latter figure.

After the rig is erected a pit five or six feet square is sunk to bed-rock, and a wooden "conductor" box made of two-inch plank eight inches square inside, is set perpendicularly between the bed-rock and derrick floor, being stepped down a few inches into the solid rock to fasten it immovably in place.

The boiler is usually of 20-horse power and the engine twelve or fifteen horse power, reversible and controlled from the derrick by means of the pulley *n n* which is geared to the throttle-valve *l l* by means of an endless cord. Another small rope *q q*, also operates the *reversing link p p*, which drops back by its own weight.

The band-wheel communicates motion to the walking-beam through the pitman *q*, and the length of stroke is regulated by adjusting the wrist-pin *p* in any one of the several holes in the crank *o'*. It also communicates motion to the bull-wheel through the bull-rope *r r* in running up the tools, and to the sand-pump reel by the friction pulley *w* while sand-pumping. This latter motion is effected by the lever *v*, joined by *u* to *t*, thus throwing the beveled friction pulley *w* into gear with the band-wheel. As the sand-pump descends by gravity a brake-block is provided behind the pulley *w* to check the speed at will by reversing the lever *v*. The sand-pump line, a light $\frac{7}{8}''$ rope, is coiled on the shaft *x* and passes up over the sheave *i i*.

The sand-pump is commonly a plain galvanized iron cylinder with a stem valve at the bottom, or is made of wrought-iron casing and provided with a plunger, not differing materially from the old style "sludger."*

The drilling tools are shown on Page Plate No. 5.

The effective cutting blow of the tools is given by the weight of the *bit*, *auger-stem* and lower link of the *jars*.

The "*jars*" are like two large flat chain links. Their function is to give the *auger-stem* and *bit* a decided *jar* on the *up-stroke*, to loosen the bit in case it should wedge fast in the hole. The *sinker bar* is added simply to give the required force to the *upward-jar*. It is never allowed to *pound* upon the drill.

The *temper-screw* connects the rope and walking-beam and by it the tools are slowly lowered as the hole deepens. After drilling about four feet it is necessary to run the

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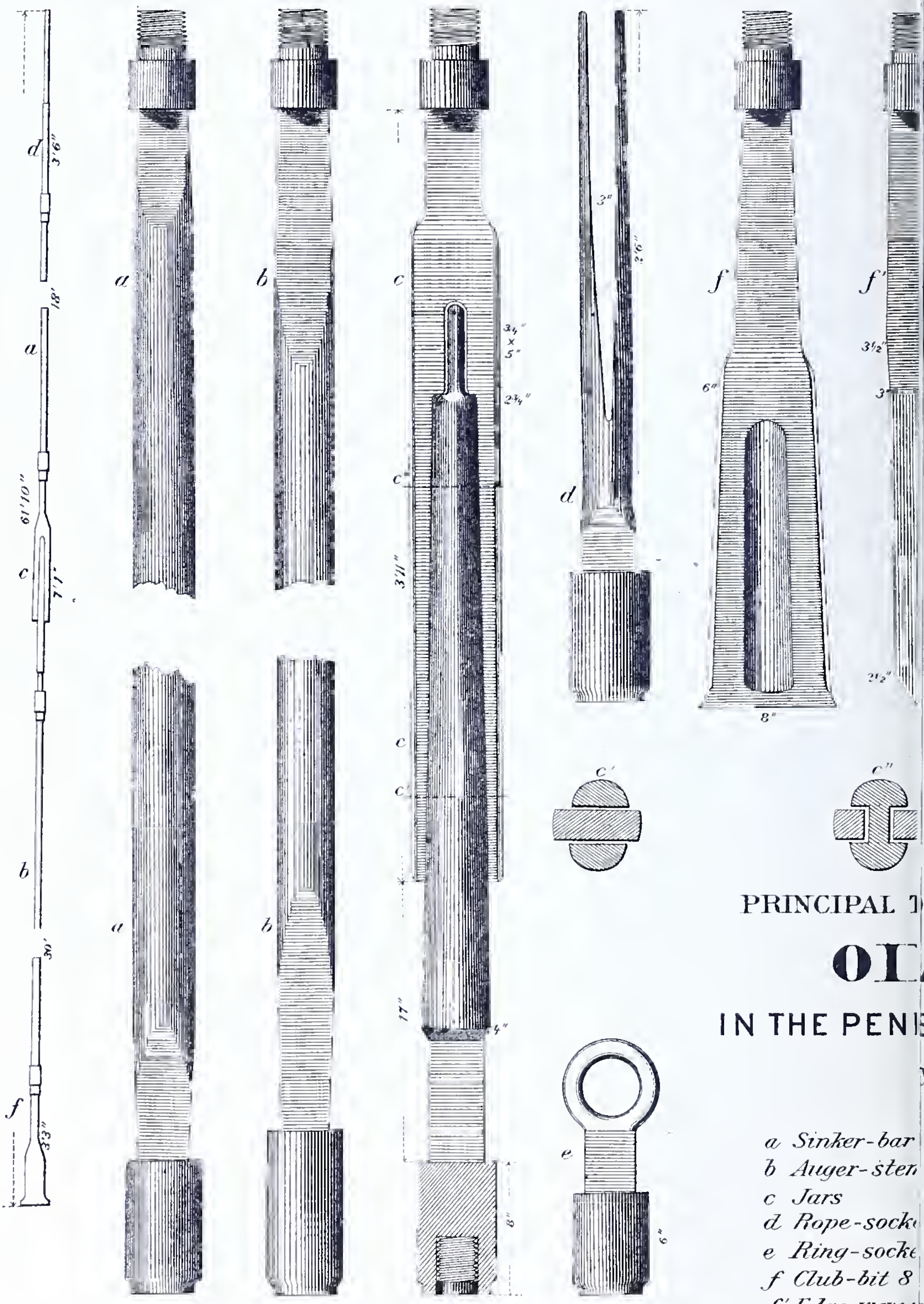
*See Ure's Dictionary—old editions.



PRINCIPAL TO

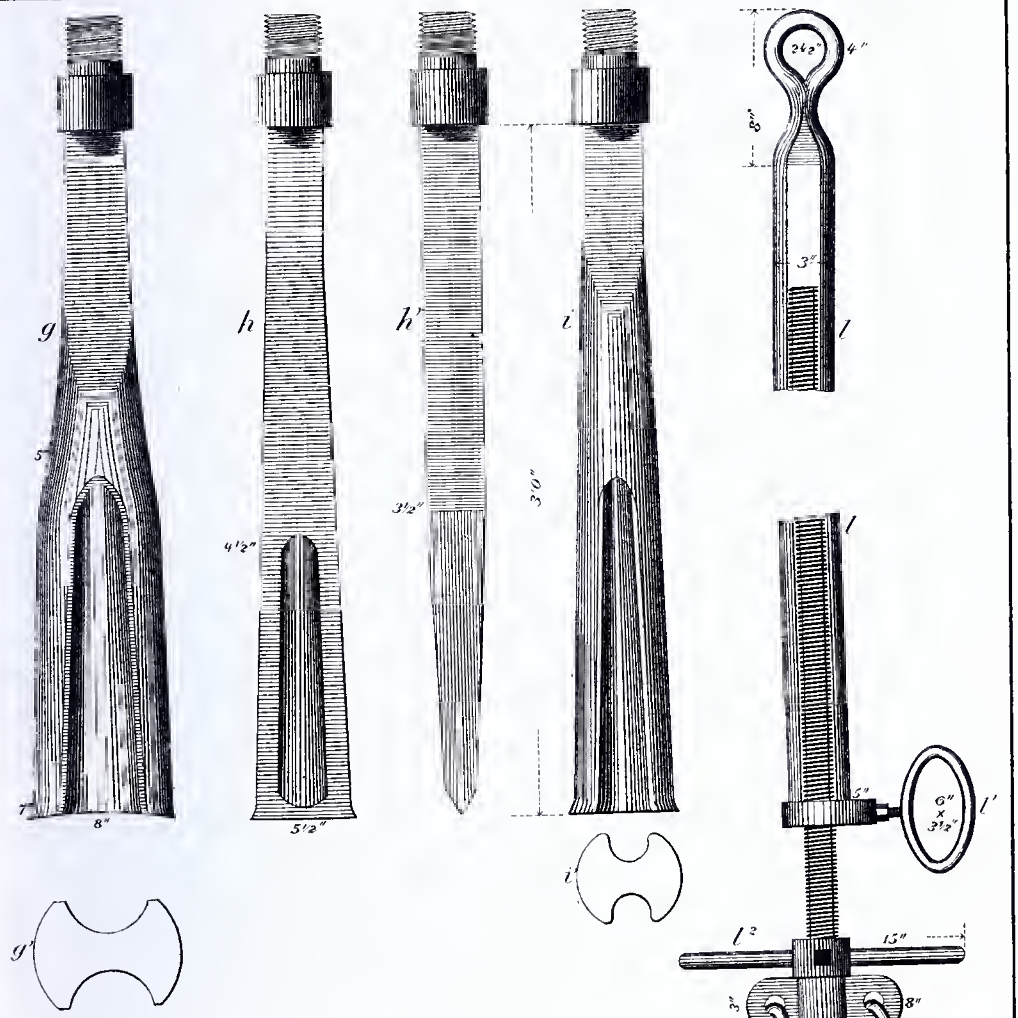
OI

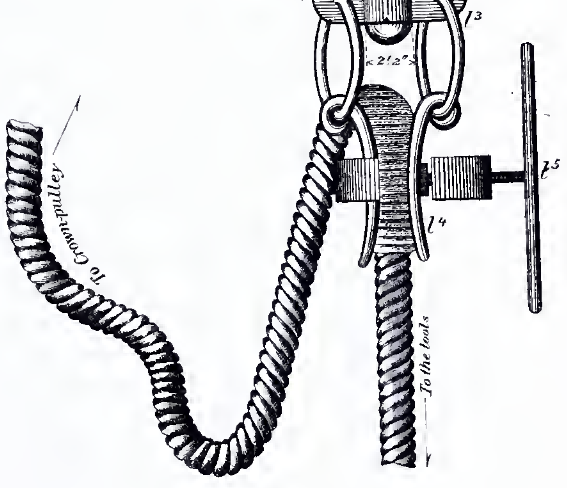
IN THE PENNSY

y

a Sinker-bar
b Auger-sten
c Jars
d Rope-sock
e Ring-socke
f Club-bit 8
f' Edge view sme
g Reamer 8"

Report A. C. Page Plate No. 5.





OLS USED IN DRILLING

WELLS

SYLVANIA OIL REGIONS

H.M.Chance.

g' Bottom view same
h Centre bit 5½
h' Side view same
i Reamer 5½
i' Bottom view same
l Temper screw
m Wrench
c'c" Sections of jars, C.

screw up and reclamp the rope, and while doing this the sediment is usually sand-pumped, and a newly dressed bit adjusted to the auger-stem.

The length of a set of ordinary drilling tools is about 62 feet and its weight 2100 pounds, thus:

	Lbs.	*Ft.*	*In.*
Rope-socket,	80	3	6
Sinker-bar, 3½'',	540	18	0
Jars, 5½'',	320	7	4
Auger-stem, 3½'',	1020	30	0
Bit, 5½'',	140	3	3
	2100	62	1

Besides these the following additional tools are used:

	Lbs.
Jars, 8'' hole,	565
Bits, (2,) 8'' hole,	320
Reamer, 8'' hole,	180
Bits, (2,) 5½'' hole,	280
Reamer, 5½'' hole,	140
Ring-socket,	50
Wrenches, (2,)	210
	3845

The cost is about $417 for the single set for 5½-inch hole, and about $275 additional for the jars, bits, and reamer for boring an 8-inch hole.

Norway iron is used for the pins, hammered iron for collars and boxes, and the stems are made of the best rolled iron.

Bits and reamers are made of Norway iron, with large steel tips, constituting about one half the tool.

"*Spudding.*"—The first sixty feet cannot be drilled in the ordinary way. The method used is called "spudding." The auger stem and bit are attached by the rope socket to a short piece of cable, (150 to 160 feet long,) the other end of which is passed over the crown-pulley and down to the bull-wheel, and a few turns taken around the bull-wheel shaft. The bull-wheel rope is thrown over, and the engine started. One of the drillers stands near the bull-wheel, with the loose end of the cable in his hands. A slight pull on this tightens the loose coils on the bull-wheel shaft, which is rapidly revolving; the tools are raised, the rope is immediately slackened, the tools drop in the hole, another slight pull is given, and the process thus goes on until a sufficient depth is attained to enable the drillers to replace this motion by that of the walking-beam.

When the bed-rock lies at a considerable depth beneath the surface, the wooden conductor is replaced by wrought iron drive-pipe. This is driven by a mall, which is raised and dropped upon the pipe, just as the tools are handled in spudding.

When the hole is sufficiently deep to admit a full "string" of tools, the drilling cable is passed over the crown-pulley and coiled upon the bull-wheel shaft, and the other end riveted into the rope socket. The tools are then attached, the joints being carefully screwed up, the bull-wheel rope thrown off, and the tools lowered into the hole by the bull-wheel brake, *c c*.

The walking-beam connections are then made with the band-wheel, by slipping the pitman on the wristpin and driving up the key; the temper-screw is suspended by a hook to the walking-beam, the walking-beam is thrown down by placing the pitman on the upper dead center, the slack in the cable is taken up, (by turning the bull-wheels), until the jars are about two-thirds open, and the temper-screw is then clamped to the rope. The bull-wheel is then given several backward turns, to slack up the cable, and the tools are then in position, ready for drilling.

As the tools rise and fall with the walking-beam, they are constantly rotated by hand by a short lever, inserted in the rings of the temper-screw; and the tools are slowly lowered as the hole deepens, by letting out the temper-screw. The driller relies solely upon the vibration communicated through the rope from the jars for his knowledge of how the tools are working.

When the length of a screw has been run, or the drill has been dulled by hard rock, the tools are drawn. The slack rope is first cleared of kinks, the bull-rope is then thrown in gear, and the bull-wheel brake applied and the engine stopped, just when all the slack rope is taken up. The clamps are then loosened, the pitman thrown off, and the walking-beam thrown up out of the way. The tools are then run up, but are stopped when the bit reaches the level of the derrick floor, where it is loosened by large wrenches, 3′ long; the tools are then run up clear of the hole, the bull-rope thrown off, the brake applied, and steam shut off. The tools are drawn aside and held by a hook, and the bit removed and replaced by one newly dressed. While this is being done, the sand-pump is run up and down once or twice, by means of the friction-gear and brake-block, already described.

After cleaning out the sediment in this way, the tools are again swung over the hole and the bit tightened up. The tools are then run down—controlled by the bull-wheel brake—the walking-beam connections made, the temper-screw clamped to the rope, and another "run" is commenced.

The average cost of *drilling* a well in the oil country in 1878 was as follows:

"Rig," complete, ...	$350
Belt, bull-rope, telegraph, water pipes, and steam fittings, ...	100
Boiler (20 H. P.) and engine (15 H. P.,) on ground,	750
Contract for drilling, drillers to furnish fuel, tools, cable, sand-pump, and line, &c., @ 65c. a foot, 1500 feet, ...	975
Total cost of plant and drilling, ...	$2,175

As prices were at their lowest ebb in 1878, this estimate should probably be increased by about 20%; say to $2,500, to make it available for comparison with other methods at the present time [1882.]

Cost of plant and drilling, ...		$2,500
Value of old rig, ...	$225	
" " fittings, ...	50	
" " boiler and engine, .	400	675
Net Cost of 1,500-foot *prospecting hole*, ...		$1,825

The *average rate of drilling*, in the Butler county oil field, where the rocks are much harder than in the McKean district, ranges from *twenty-five* to *fifty feet* a day (twenty-four hours); but sometimes as much as a hundred feet is drilled in one day, under favorable circumstances, while a very hard "shell" one or two feet thick may require twenty-four hours of hard work.

The cost of drilling prospecting holes by this method in the anthracite regions will be very much greater than that shown by the above figures, because the rocks are much harder, and are inclined at considerable angles from the horizontal.

The high inclination of the rocks increases the risk of flat and crooked holes, and necessitates careful watching and constant and uniform rotation of the drill. In some districts, as, for instance, the Wyoming basin, the hardness of the rocks is not a serious obstacle, but will probably reduce the average drilling speed to less than one half that attained in the oil regions. This will add materially to the cost. We have not sufficient data on which to base general conclusions as to the cost per foot of rope drilled holes, but it certainly seems probable that they can generally be sunk much cheaper (for deep holes) than by the diamond-drill process.

This process (rope drilling) has also been used for drilling holes for steam and compressed air pipe, for pump-holes to replace column pipe, for artesian wells, (as is being done at present at Nesquehoning Tunnel,) and for holes used as ropeways, for conducting an inside slope rope to a winding engine at the surface. A hole 600 feet deep has lately been drilled for this purpose by the Lehigh and Wilkes-Barre Coal Company at the Hollenback Colliery.

Percussion Power Drills.

Although for limited depths percussion drilling machines may be used for prospecting, the principle on which they are constructed disqualifies them from the work of deep boring. They have not sufficient drawback power to lift a long line of rods in a vertical hole, and in a horizontal hole the impact of long rods is ruinous to the machinery. About the only prospecting that can be economically done with them, is the drilling of holes of limited length into unknown ground from the galleries of a mine already opened.

The Diamond Drill.

This drill has been largely used for prospecting purposes in the anthracite regions, and generally with good results.

The claim frequently made that *complete cores* can be obtained showing the *exact condition* of a coal bed, cannot be entirely substantiated, for the cores so obtained are not infrequently very imperfect and are often little if at all better than loose fragments, and at times a considerable portion of the core is completely pulverized and lost.

It is also claimed that the cores show the dip of the rocks, but as the rocks may be *false-bedded* or may exhibit a decided cleavage not coinciding with the bedding, it is not safe to base any important conclusions on deductions based alone on the apparent dip shown by the cores.

The cost of drilling deep holes with the diamond drill in rocks of moderate hardness is certainly much greater than by the rope-drilling method as practiced in the oil regions, and the progress made is also much slower,—the cores obtained may or may not be worth the difference in time and money.

When a large number of comparatively shallow holes are to be drilled in hard rocks ; or prospecting holes are needed from some point inside a mine ; or when inclined or horizontal holes are required, the diamond drill will give the best results.

The "long-hole" method of shaft sinking by the diamond drill will be considered in another chapter.

This drill is so will known among mining men, and it has been so thoroughly illustrated and described in various publications, and by circulars issued by the company, that it has not been considered necessary to illustrate it by engravings or to dilate upon the details of its construction and operation.

When a large number of holes of moderate depth are drilled by a corporation owning and operating the drill, the cost per foot may, by proper management, be much reduced. Mr. Lewis A. Riley* reports the average cost of 24 bore holes (ranging from 100 to 900 feet in depth) aggregating 9902 feet, at $2.22 per foot. Their average depth was about 400 feet. The *average* progress made in ten hours was 18.9 feet. The above estimate of $2.22 per foot covers everything except royalty, interest on investment and wear and tear, for which 20 or 30 cents should be added.

For holes double this average depth of 400 feet the cost per foot would probably be nearly if not quite doubled. Mr. Riley divides the cost thus:

Labor,	$1.15
Diamonds,	.66
Fuel, water and repairs,	.41
	$2.22

In the transactions of the American Institute of Mining Engineers, Vol. II, page 241+, Mr. Oswald J. Heinrich gives figures that show how rapidly the cost of drilling by this method increases with the depth. The boring was done at the Midlothian collieries in Virginia.

Depth of Hole.	*Cost per foot including Interest.*
419 feet,	$1.86
850 " 3 inches,	2.43
1142 "	3.62

His estimates were based on the following scale of wages for a twelve-hour shift:

Skilled labor, such as foreman, etc.,	$2.50 per day.
Engineer,	2.00 " "
Assistants,	1.50 to 2.00 " "
Laborers,	1.00 " "

The cost per foot for labor was,—

For 419-foot hole,	$ .36
For 850 " "	.59
For 1142 " "	1.02

Constructing Geological Cross-Sections.

The practical value of data obtained from surface examinations, prospecting holes, entries and shafts is greatly enhanced by properly constructed cross-sections and maps, and these are always serviceable in planning future developments.

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*Transactions American Institute of Mining Engineers, Vol. V, page 306.

When all the outcrops with their dips, the prospecting entries or shafts and the bore-holes, have been carefully located on the map an attempt may be made to draw lines connecting the isolated outcrops, to represent the approximate position of the coal blossoms on the surface.

To avoid the possible error of connecting outcrops on two different seams the number and order of the coals is first determined by a comparison of local sections compiled from exposures noted in making the surface examinations.

The course of the outcrop line of any layer in stratified deposits is governed by the dip of the stratum and the contour of the surface. When the dip is vertical the surface contour does not affect the course of the outcrop which is then coincident with the "strike"*; in horizontal seams the outcrop follows every irregularity of the surface, accurately conforming to the curvature of a water-level surface contour line. Between these two extremes of the vertical and horizontal dips, all degrees of gradation, are found so that it is often a rather difficult problem to determine how much an outcrop line will be deflected to one side or the other by surface irregularities.

Probably the most trustworthy plan is to trace the outcrop of some prominent stratum and use its position as a key to the course of other outcrop lines.

When a contoured map of the surface can be obtained the location of the outcrop lines is greatly simplified; but the construction of such a map generally involves more labor than can be expended upon it. A rude sketch contour map can sometimes be made at the same time with the surface examinations, that will be serviceable, even if only approximately correct.

Such a map will show the trend of the hills, which usually nearly coincides with the strike of the rocks, the best route for a railroad or wagon road, and many other valuable facts may be developed by it, for as topography and geology bear such intimate complementary relations, a study of the one must always throw some light upon the other.

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*"Strike": the direction of a water level driven in the bed.

For the construction of a cross-section, the line of section is best located so that an equal number of the exposures to be utilized will be found on each side of it ; the error caused by projection is then reduced to a minimum. All the exposures of rock or mineral are then projected upon this line and a profile constructed showing them at their proper elevations above an assumed or known datum plane.

When the dips are slight, the elevations are of great importance, and are determined as carefully as possible. If the same rocks are identified on both sides of the property and the dip of the rocks is slight, there is usually little difficulty in completing the section. When the dips are steep and the structure is complicated by anticlinal and synclinal folds, the construction is much more difficult.

An exact method of determining the underground curves from surface dips cannot be devised, and in the absence of exploratory bore holes, the depth and curvature of a coal or other mineral bed can be approximately indicated only, and by very imperfect methods.

I have been unable to find any publication containing a discussion of this subject. In some of our technical schools and colleges the student is required to construct or copy (generally the latter) a number of profile sections as part of his routine work, but little attention is given to the methods of projecting observed dips and exposures upon the section line, and still less to the method of constructing the under—(or above)—ground curves. As a result of this system of instruction every engineer or geologist is compelled at the start to invent for himself a system, or copy the style of others.

Geological profile sections are frequently used to determine underground structure from data obtained at the surface, but the experience of engineers in the anthracite coal-fields has shown them to be reliable in exceptional instances only.

The possibility of indicating underground curvatures from dip-angles observed at the surface has been found to be limited to a mere approximation of the larger flexures. In the anthracite coal-fields the hypothecated structure in undeveloped ground is usually introduced upon the working profiles in pencil lines only, and as development progresses frequent changes are necessitated by the eccentricities of individual seams.

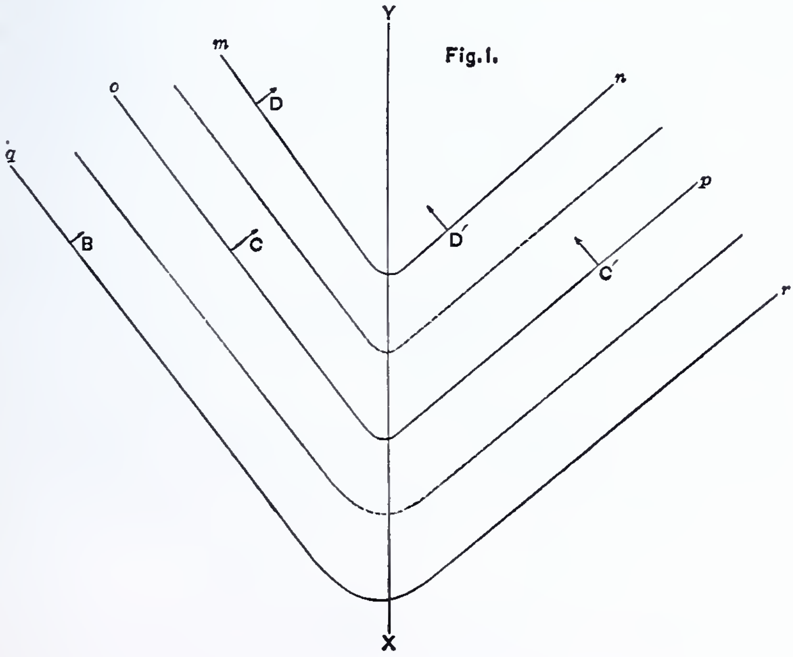
It will be in place here to describe the methods commonly used in the construction of cross-sections, and to indicate some of the reasons why these must always fail to give trustworthy results.

We may consider these points under three heads:

1. The location of the section line.
2. The projection of dips and exposures.
3. The construction of underground curves.

Having given a series of exposures and dip-angles located geographically upon a map, the best position for a section is generally conceded to be along a line drawn midway between these exposures, and at right angles to the line of strike, *i. e.*, parallel to the direction of the dip.

If the section line crosses an anticlinal or synclinal axis decidedly inclined to the horizon—that is, when the opposing dips are not parallel, and the lines of strike on opposite



sides of the axis converge—the section may often be best located along a broken line, the lengths of which are drawn at right angles to the local strike; or, it may be constructed on a line drawn at right angles to the axis, when this bisects the flexure, or on a line drawn at right angles to a line bisecting the angle formed by the converging strike lines.

Thus, in Fig. 1, three coal beds, B, C, D, are exposed at the points, B, C, D, D′, and C′, dipping as indicated by the arrows; the line XY is the axis of the flexure, rapidly deepening towards Y. The lines *mn*, *op*, *qr*, are water-level contours drawn through the different seams, representing, therefore, their "line of strike." The dotted lines are introduced to show the method of locating hypothecated horizons.

The best method of constructing a section through these exposures is illustrated by Figs. 2 and 3, in which the section line, GHI, is a broken line, the segments of which, drawn parallel to the opposing dip-angles, intersect on XY.

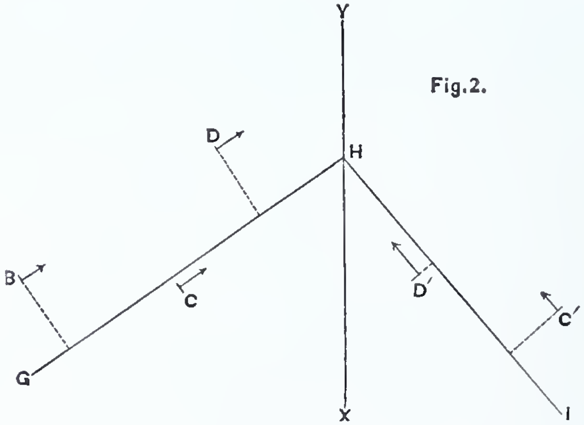
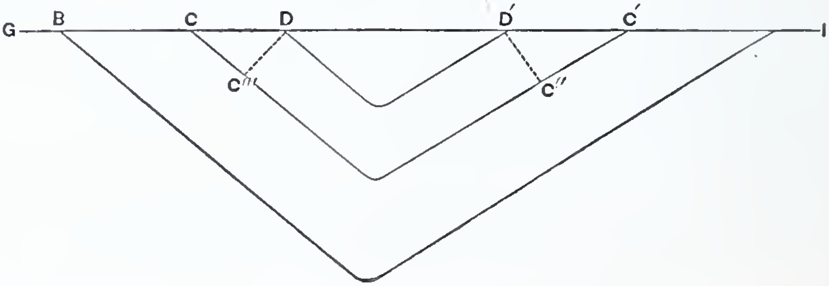
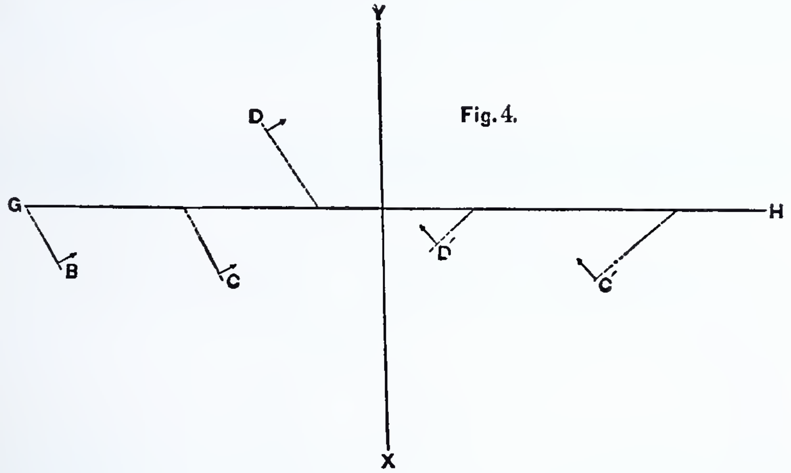


Fig. 2.

Fig. 3.

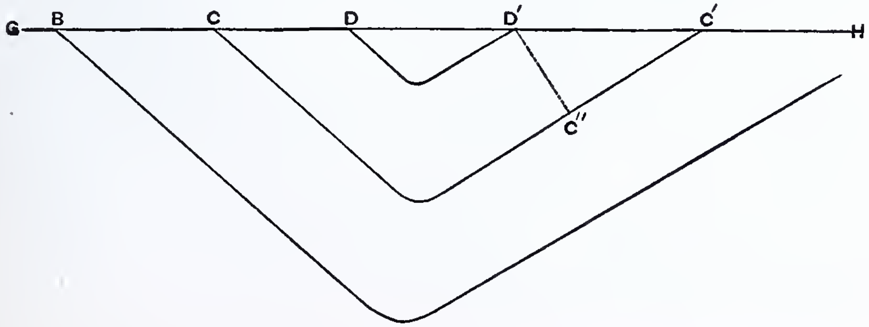


The resulting section (Fig. 3) shows the absolute thickness of the strata between beds B and C, and between beds C and D, and the agreement between the thickness DC‴ with that determined at D′C″ furnishes a good check on the accuracy of the construction.



Two very objectionable methods are shown by Figs. 4,5 and 6. The section line GH is drawn at right angles to the axis XY, and midway between the located exposures. The first section, Fig. 5, is obtained by projecting the dip-angles upon the section line, *in prolongation of the line of strike*, as indicated by the dotted lines in Fig. 4. This is the only proper method of projecting exposures, and gives a section approximately correct for the adopted alignment, but the thickness of measures as represented by the section is much greater than the true thickness. This can be seen by comparing D′C″ in Fig. 5, with the same line in Fig. 3. When

Fig. 5.

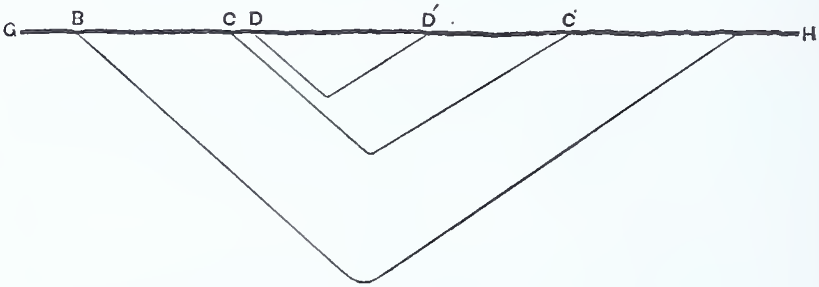


this method is adopted, a prominent "N. B." should always direct attention to this exaggeration.

Fig. 6, shows the absurd construction that results when the dip-angles (exposures) are projected at right angles to the section line—a plan often thoughtlessly adopted.

Difficulties in the identification of coal seams on opposite sides of a basin may frequently arise from distortions of this kind.

Fig. 6.



In the construction of underground curves, two methods are in general use:

1. By equal thickness.
2. By equal (parallel) dips.

The calculated thickness of each stratum is rigidly preserved throughout in the first method, by making the curvature in each seam concentric to those above it. These curves are usually compound curves composed of circular arcs.

When the second method is employed, no attempt is made to preserve uniformity of thickness along the axes of anticlinal or synclinal flexures, the opposing dips are prolonged until they intersect, and the curvature sketched in to suit the fancy of the draughtsman; but a combination of this and the preceding method is sometimes employed.

The difference between these two methods is illustrated by Figs. 7 and 8, in which the surface observations are identical, but the underground (hypothecated) curves vary widely. In Fig. 7 each stratum preserves a uniform thickness throughout, but in Fig. 8 the prolongation of the opposing dips produces an augmented thickness along the synclinal and anticlinal axes.

Fig. 7.

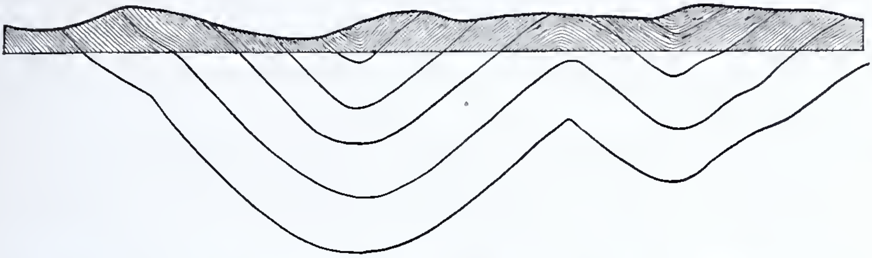
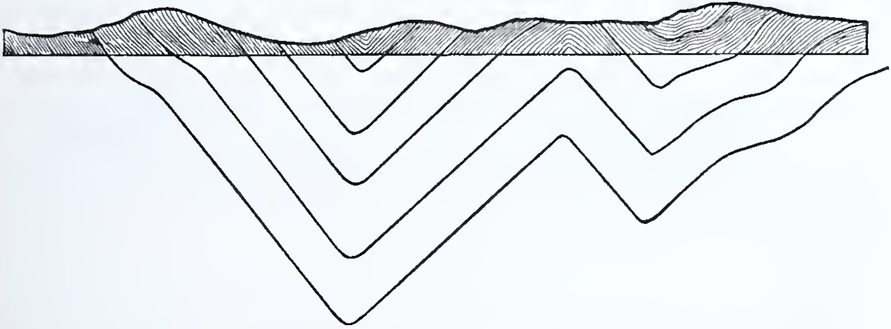


Fig. 8.



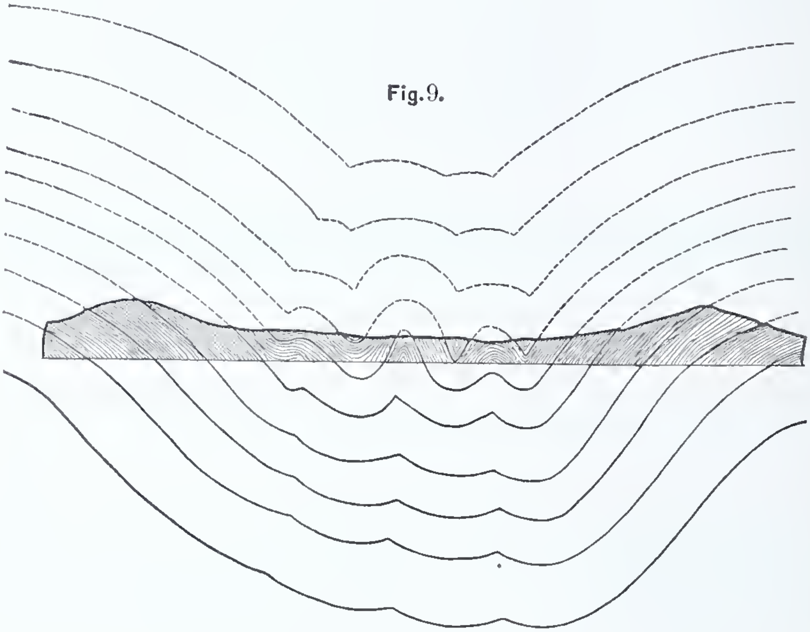
As it is evident that the first method will show the minimum depth of any coal in the basin, and the second the probable maximum depth—in all but exceptional instances—a combination of these two methods is apparently preferable, as more nearly approximating the probable underground curvatures. Such a combination has been employed by some members of the Geological Survey in the construction of many of the sections in Pennsylvania.

But there is a serious objection to any system involving in whole, or in part, the method of equal thicknesses, viz: *the gradual obliteration of the surface flexures in the underlying rocks.*

This is illustrated by Fig. 9, which is constructed partly on this system. It will be observed that the steepest dips are *along the line of observed exposures*, (*the surface;*) above and beneath this line the axes rapidly fade away, and can hardly be seen in the basal stratum. The same result obtains in the (restored) formations lying above the present surface.

Sections constructed on this principle (Figs. 7 and 9) will, therefore, always show the maximum curvatures along the line of observed dips. It is hardly necessary to say that any method uniformly giving these results must be essentially false. The regularity of the curves in profiles constructed by this method is often very misleading.

Fig. 9.



Under *one sequence of conditions only* does this method give results approximating the true underground structure. Having reference to Fig. 9, let us suppose the lower strata consist of hard, massive sandstones, or conglomerates, and the upper rocks of soft slates or shales. If a lateral compressing (flexing) force acts upon such a series, the lower massive formation will yield in curves of large radii, while the overlying softer strata, compressed in the trough thus formed, will be thrown into a series of crimples resulting in flexures similar to those shown in the section.

If, on the contrary, the upper rocks are hard and are underlaid by softer rocks, these latter would suffer the greatest contortion, whereas the construction shows them nearly flat.

I have lately made a series of experiments on rock contortion, by artificially compressing layers of clay, paper, india-rubber, wax, etc., hoping to be able to deduce some laws governing the relationships of the resulting curves, which might be practically applied to geological sectioning; but the factors involved are so numerous and vary between such wide limits that it seems an almost hopeless task. I have, however, arrived at some general conclusions that may be of service.

1. The subordinate flexures or sub-basins in a synclinal of soft rocks underlaid by a hard resisting formation, decrease in sharpness towards this basal rock.

2. If a series of soft measures underlies this hard formation, it may be contorted by a series of flexures bearing no relation to the upper series.

3. When large masses of approximately uniform hardness are contorted by compression, the resulting curves bear close complementary relations to each other. (See Fig. 8.)

The first of the above propositions has already been referred to on a preceding page. When these conditions obtain, as in the anthracite coal basins, the coal-measure rocks are crimpled and distorted while their hard basal plate of conglomerate has been arched into curves of large radii, with flat dips, as proved by diamond drill-holes. When these conditions obtain, the method of equal thicknesses, or a modification of that method (Figs. 7 and 9), will give curves more nearly approximating the underground structure than any other method.

On the Lehigh River we find a striking exemplification of the second proposition. At Slatington the Lower Silurian (roofing) slates, No. III, are twisted and contorted by a complex series of overturned flexures. Above these come the hard sandstones and conglomerates No. IV, forming the mountain, dipping down in curves of large radii beneath the softer Upper Silurian strata of Hazardville and Bowmansville. These latter are twisted, faulted, and overturned, but these flexures can bear no possible relation to those in the roofing slates, lying five thousand feet beneath, and separated from them by the hard rocks of No. IV, which reflect the minor flexures of neither the former nor the latter. Under conditions similar to these it is of course utterly impossible to indicate the curves of the lower formations from data obtained from the upper rocks.

The conditions of the third proposition are strikingly shown by cross-sections of the coal measures in some parts of France, in which the various coal seams are bent in a series of parallel zig-zag lines. Here the second method, *i. e.*, by equal dips, is peculiarly applicable.

When the observed dips are in a hard, massive formation underlaid by softer rocks, the underground curves are always indeterminable.

Too much reliance is often placed upon geological cross-sections as a means of determining underground structure. In this their utility is limited, but as a means of estimating rock thickness, as a help to intelligent prospecting, and as a pictorial means of representing demonstrated and hypothecated structure, their value is great.

A series of crimples may be present in the trough of a synclinal flexure which has no representative at the surface. In such a case a number of bore-holes judiciously located across the basin will generally furnish data from which a section closely approximating an accurate representative of the structure can be constructed.

Geological cross-sections may also be of value in locating the position of an outcrop at a point where it seems advisable to drive in the main drift of projected workings ; but they are more frequently employed in determining the best location for projected shafts or slopes.

The policy of thoroughly checking the structure indicated by geological cross-sections by a series of bore-holes, prior to the location of deep shafts in undeveloped ground cannot be too strongly urged. Instances are not wanting in which large expenditures might have been saved had this method been more generally adopted ; and these will probably lead to its universal adoption in future experimental enterprises in the anthracite regions.

Samples for Analysis.

In selecting a sample for analysis, the object to be attained is to secure a sample fairly representing the average quality, not of the coal in the bed, nor of hand specimens, but *of the article to be shipped to market.*

When mining and shipping are already in progress, the most approved plan is to take a considerable quantity (two or three shovels full) from a number of cars, or to gather it in buckets from the loading shute.

A large quantity having been thus collected, it is carefully broken down into smaller pieces, (on a clean floor,) thoroughly mixed with a shovel, and thrown up in a conical heap. This heap is then quartered, and two quarters opposite each other thrown aside. The remainder is broken still finer, mixed, heaped up and quartered as before; and this process is repeated until only a few shovels full—about enough to fill a small keg—is left. This is packed and shipped to the chemist, the box or keg being lined with strong paper, to prevent the loss of the fine material and the entrance of dirt.

It is almost unnecessary to urge the importance of intrusting this matter to an entirely disinterested person.

If no coal has been mined, and the sample is to be taken from a drift, tunnel or slope sunk on the bed, a small piece should be broken from *every inch of the seam*, care being taken that the fragments shall be of equal amount from each part of the bed, so that they will make such a sample as would be obtained by cutting out a block of coal three or four inches square, (or less,) and as high as the bed is thick.

The parting bands of slate, sulphur, (pyrites,) etc., should always be included in the sample, unless the bed contains a thick layer of slate or bone that readily breaks loose from the coal, and can be cleaned from the coal in mining. In this case the parting may be represented in the sample by such a piece as would be taken if the parting were only a thin layer.

As very few persons can be intrusted with the selection of samples taken directly from a bed not mined, when work of this character is needed, and the analyses are to be used for any important purpose, a mining engineer accustomed to work of this kind should always be employed to gather the samples.

Estimates of Area and Tonnage.

Estimates of the tonnage or amount of coal in the ground should be made by the acre—not of coal land, but of the actual acreage of the bed itself.

As the available or contained tonnage varies with the specific gravity of the coal, it is not possible to give any fixed figure to represent the amount of coal per acre for each foot in thickness.

The specific gravity of anthracite varies widely, (see Chapter I,) but for rough and ready use we may assume an average of 1.5 to 1.6, and for Bituminous coal an average of say 1.4.

On the former basis an acre of Anthracite coal one foot thick will contain in round numbers about 1775 tons, and of Bituminous coal about 1695 tons ; a cubic yard of Anthracite will weigh about 2530 ℔s. or 1.13 tons, and of Bituminous about 2360 ℔s.=1.05 tons*.

In measuring the coal-beds for estimates of total contents all slate bands *must be excluded*, but as some thin partings ("*leaves*") and bony coal are nearly always included in the measurement, it is well to make a certain deduction,—at least 5%,—to guard against exaggeration.

When geological sections have been prepared showing the approximate depth of the basins, and a geological map defines the limits of the bed or beds, the calculation is a simple but tedious process.

Deductions are made for soft outcrop coal not of marketable quality. When the bed is bituminous and lies flat with little covering, a margin sufficiently wide to insure at least twenty feet of cover should be deducted from the actual acreage.

In steep-pitching anthracite coals an allowance of eight or ten yards (and often much less) will generally include all the soft coal.

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* Of 2240 ℔s.

CHAPTER III.

Methods of Opening Coal.

In the selection of the best means of developing a coal tract, the geological structure of the underlying rocks exercises a governing influence.

When the coal outcrops within the limits of the property it is usually opened, if flat, by a water-level drift, but if highly inclined by a slope sunk directly down upon the seam, or by a tunnel driven across the intervening measures.

In the bituminous coal areas where the seams lie very flat, dipping from ten to one hundred feet to the mile, the water-level drift is almost universally used, but shafts of moderate depth are not uncommon in some districts. To insure free drainage, the lowest accessible point upon the outcrop is chosen as the best location for the main entry, thus at the same time securing a favorable grade for the haulage of the loaded mine cars. As the outcrop coal, as described in the preceding chapter, often dips *into* the hill, the drift is usually commenced a few feet below the coal terrace, and driven on a slight up-grade until the normal dip is reached. It sometimes happens that this inward dip is so strong that it is advisable to open by a shaft sunk in the center of the basin,—provided that the depth of such an opening is not too great and the amount of water to be pumped comparatively small. When the inward dip to the center of the basin does not exceed twenty-five feet, drainage may be accomplished (through a drift) by means of iron pipes, two, three, four or more inches in diameter, running from a *sump* in the center of the basin to a point outside the mine at a lower elevation. These pipes when filled with water form a self-acting syphon and will deliver a large quantity of water. Stop-cocks or valves are placed at each end of the pipe and a force-pump attached to one end to start the flow by filling the pipe with water, an additional cock being placed at the highest point on the pipe to draw off air that may accumulate at that point.

In the anthracite regions the water level drift is used in opening inclined seams exposed in ravines or gorges eroded across the strike of the measures. It is then similar in every particular to a gangway driven underground from a shaft or slope. As drifts are driven and timbered precisely like underground gangways, they will be described in the chapter relating to that subject.

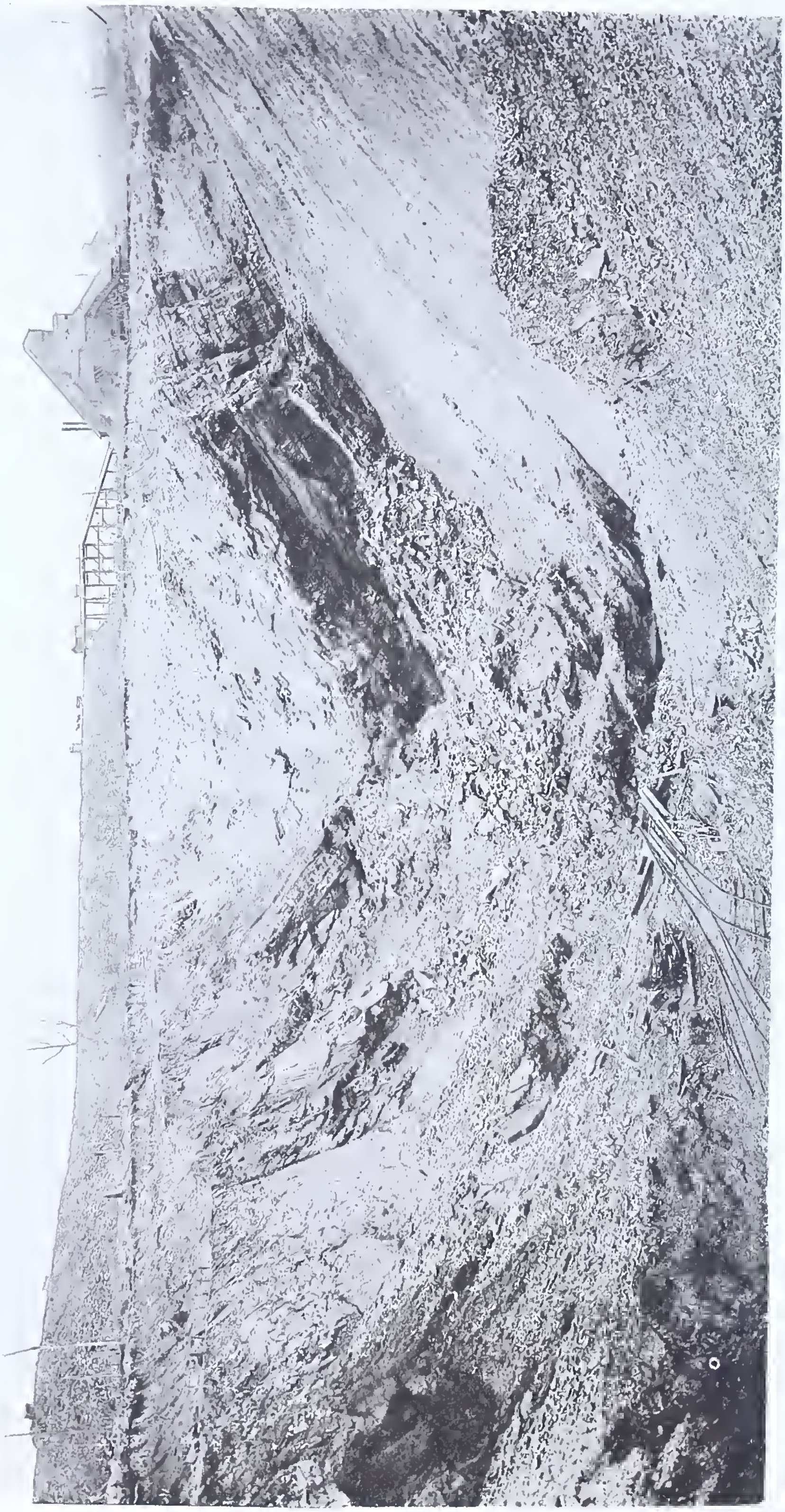
When the seam is inclined, and is accessible at no point along its outcrop low enough for the location of a drift that will command sufficient breasting area, it is opened by either a slope or tunnel.

Seams dipping *with* the slope of the ground can generally be reached by tunnels of moderate length; but when the dip is *away* from the hill the length will be much greater, and, unless several beds are to be opened and worked, it has seldom been found advisable to open in this way. Tunnel collieries have been found to be much more cheaply operated than either shaft or slope openings, as neither hoisting or pumping machinery is needed. But the grave fact that when the coal above water level is exhausted the tunnel becomes almost worthless, is a serious offset to these advantages.

Tunnels are most extensively used as a means of reaching beds underlying or overlying a seam in which workings have already been opened, by driving underground directly across the intervening measures. Such a gallery, strictly speaking, is not a tunnel, for a tunnel is a passage driven from day to day, and open at both ends; but custom sanctions this use of the term in the anthracite regions.

When the coal does not come to the surface within the area to be developed, it is generally necessary to open it by shaft, but under certain conditions the opening may be made either by slope or tunnel.

A slope opening may be made when the coal is brought to within a moderate distance of the surface by an anticlinal



STRIPPING AT HOLLYWOOD COLLIERY NO. 1,— LOOKING EAST. (PICTURE REVERSED.)

ARTOTYPE E BIERSTADT, N Y

axis, as at the Shenandoah City Colliery. The upper part of the slope is there sunk through rock (a "Rock-slope") and should therefore be called an *incline** rather than a slope, but this distinction is not recognized by the miners.

A tunnel opening is sometimes made when the coal, dipping steeply in the same direction as the slope of the hillside, outcrops on an adjoining property; or when it turns over forming a saddle in the hill.

Nearly all the early mining operations were prosecuted through water-level openings,—drifts and tunnels,—driven in from the surface and giving access only to the coal lying above water-level. The amount of coal now accessible in this way is comparatively small, for these old workings exhausted or ruined portions of the best and thickest coals lying above water-level.

Slopes are greatly preferred to shaft openings at all points in the anthracite regions where the coal is accessible along its outcrop and where the dip is more than 15° or 20°.

The usual practice is to sink the slope and its air-way side by side, and at a depth of from one hundred to one hundred and ten yards on the bed to open out a "lift" by driving gangways to the right and left. The breasts opened along these two gangways (east and west) constitute the "First Lift." This is often worked for several years before the slope is sunk to open a second lift, but it is now frequently the practice to continue the sinking without waiting until the coal on the upper level is nearly exhausted. When this is done, work on the slope is prosecuted at night,—at least the raising of rock and coal and lowering of supplies,—so that this work does not interfere with mining and raising coal from the upper level.

The "Second Lift" is usually opened out at a depth of from seventy-five to one hundred yards from the first lift.

Sometimes three or four lifts are open at once, but there are few collieries mining coal from more than three lifts at the same time.

In "lift" mining the water is pumped from each of the different gangways, and at many collieries nearly all the water is caught on the first and second lifts ; the lower levels being very dry. The surface water is often caught on the water-level gangway or drift, driven between the outcrop and the first lift, and conveyed directly from the mine without pumping.

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*See Glossary.

When the coal is to be opened by shaft, it is generally considered best to locate the main opening directly over the deepest part of the basin found on the tract to be developed.

Before the best location can be intelligently determined it is absolutely necessary to have all the geological information attainable represented on an accurate map, and one or two cross-sections constructed to show the geological structure. The depth of the basin and its location as shown by these sections, may or may not be trustworthy, and in my opinion, it is therefore never wise to proceed until the results so obtained have been verified by a series of bore holes, especially if the proposed opening is in a district subject to marked irregularities in the condition of its coal seams. The proper location of the main winding and pumping shaft, for the most economical extraction of the coal is a matter of such vital importance that the expenditure of a few thousand dollars in preliminary exploration by boring seems by comparison a small matter.

The Pennsylvation Mines Ventilation Act requires two openings not less than one hundred and fifty feet apart. Unless the coal to be worked lies in two sub-basins, the location of the second outlet is not a matter of much importance ; but if the tract to be developed is crossed by more than one axis, the second opening is best located in the trough of the minor basin, and may then be used for raising coal and water from the area that cannot be economically worked from the first shaft. Other things being equal, it is evident that it is advisable to have the two openings some distance apart.

The coal of the future will be raised from deep mines, through shafts, and as shaft after shaft is sunk in progressively deeper portions of our anthracite basins, this subject —the proper location of shafts—will be more generally recognized as one of preëminent importance. In the Pottsville, Wyoming, Ellengowan, etc., basins, the great masses of untouched coal lie at considerable depth, and can only be worked through shaft openings.

When a shaft passes through two or more workable beds, the mining operations then closely simulate "lift mining," and in such cases nearly all the water is usually caught in the upper beds; thus at the large Pottsville shaft the water is nearly all caught in workings on an upper bed at a depth of about three hundred feet.

Chapter IV.

Shaft sinking and timbering.

Winding shafts in the anthracite mining regions are always square or rectangular; second openings used as upcast airways are occasionally round.

The surface material—earth, clay, gravel, or loose sand—passed through before reaching bed-rock is usually not more than twenty feet thick, and for this thickness an ordinary cribbing of heavy timbers (12×12 or 12×16) or a masonry curbing is generally sufficient; but when the surface material is much thicker, or when the material is loose and runs like quicksand, or is water-logged, a double curbing is necessary.

Under such conditions the shaft is often at once divided into the required number of compartments by heavy buntons alternating or placed "skin to skin," thus effectually bracing the cribbing against the lateral pressure exerted by loose material.

In the Wyoming Valley at points where a large amount of diluvial material must be passed through before reaching bed-rock, much difficulty will be experienced in the future in attempting to sink shafts to develop the deeper portions of the basin between Pittston and Nanticoke.

The deposits of clay, gravel, and sand now filling the old valley of the Susquehanna river between these points, is in place one hundred and fifty feet deep. To sink through such a mass of loose material we will probably be forced to avail ourselves of the methods now used in several of the European coal fields.

The English and French works are filled with descriptions of water-tight tubbing, moss-boxes, etc., that have been successfully used in sinking through loose and water-logged strata. It is not within the province of this report to discuss the relative merits of the different systems in use abroad, but it does not seem probable that any of these methods will be entirely successful in the anthracite regions. They will doubtless be modified to meet the peculiar conditions of each case—but these modifications must in some respects be radical, and the best results will probably be obtained by adopting, from each of the several systems, the methods and appliances that have given the best and cheapest results.

Size of Shafts.

The average size of shafts now being opened in the anthracite regions is considerably larger than in former years. This change has resulted from three causes.

1. Greater depth of shafts to open undeveloped coal in the deeper portions of the main basins, which necessitates
2. Larger cross-section, to secure better ventilation for these more gaseous mines.
3. The policy now adopted of opening one large colliery in preference to two small mines, necessitating better hoisting facilities.

The width of shafts is usually governed by the *length of the mine cars*, which ranges from 9 to 10½ feet; hence, the width adopted is usually 10, 11, or 12 feet.

The length is governed by the number of compartments, and by the area of airway needed.

The compartments are now made from 6 to 8 feet broad, inside the buntons, or an average of about 6½ feet between the guides; hence, the compartment, measured between the centers of the buntons, is usually from 7 to 9 feet.

The length, therefore, varies from 16 to 20 feet, for a shaft with two compartments, 30 to 38 feet for a shaft with four compartments (22 to 26 feet being the ordinary length for three compartments,) to 44 to 52 feet for a shaft with six compartments.

When this large number of compartments (6) is provided, two are usually used for an upcast airway, as at the Exeter shaft, (see Atlas, Plate II,) and for the pump-rods and column-pipe, and in some cases one is used with the hoisting compartments as a downcast, while at other shafts two extra hoisting compartments are held in reserve for future requirements, as at the Exeter colliery.

The shafts at present operated in the anthracite region are very rarely more than eight hundred feet deep; their average depth is probably between three and four hundred feet.

The deepest shafts are found in the neighborhood of Wilkes-Barre, and in the southern basin, near Pottsville, where shafts from six to eight hundred feet deep are not uncommon.

The dimensions and depth of some shafts in the Wilkes-Barre district are given in the following table.

NAME OF SHAFT.	OPERATOR.	Length.	Width.	Depth.*	Proposed depth.	REMARKS
Plymouth, No. 3, . .	D. & H. C. Co.	38	10	336	. . . .	Finished.
Boston, . . .	D. L. & W. C. Co.	24	12	200±	. . . .	"
Kingston, No. 3, . .	Kingston C. Co.	33	12	544	. . . .	"
Kingston, No. 2, . .	" " "	17	11	. . . .	. . . .	"
Kingston, No. 1, .	" " "	24	12	.	. . . .	"
South Wilkes-Barre,	L. & W. C. Co.	24	12	590	1100	Sinking.
Bennett, ...	Waddell & Co.	20	10	310		
Alden, ...	Alden C. Co.	26	12	30+	270±	Sinking.

* January, 1882.

Some of the shafts now under contract are of extraordinary size, notably, the Dorrance shaft of the Lehigh Valley Coal Company, 13′×52′, and the Woodward shaft of the Delaware, Lackawanna, and Western Coal Company, 53′×10′. This latter shaft was originally started 12′×60′, but trouble from loose decomposed rock made it necessary to decrease its size to the present dimensions. The Gaylord shaft of the Gaylord Coal Company is also of notable size, being 12′×47′.

In the Pottsville, and also in the Shenandoah and Mahanoy Basins, the shafts are of much smaller size. Gas is not so troublesome here, old slopes can be used as pump-, man-, and air-ways, and the shaft can generally be used exclusively for raising coal ; hence only two compartments are necessary.

When the opening is located on ground covered by a moderate depth of soil or drift material, a rectangular pit is sunk at once to bed-rock. This pit is made from four to eight feet wider and longer than the proposed shaft dimensions, and is carried down one, two, three or more feet into the decomposed rock, or until a firm substratum is reached. It is then timbered up with a solid cribbing of large square timber,—about 12×12,—which is either mortised or held firmly together by keys and gains.

This timber cribbing is often replaced by a solid stone curbing, which is for some reasons decidedly preferable, especially as it can be used as a foundation for the head-frame.

When the soil is of considerable depth and has a tendency to fill the pit by caving in, it is sometimes held in place by a temporary set of timbers until the permanent cribbing is inserted.

At large shafts the cribbing is usually braced by buntons which divide the shaft into the necessary number of compartments. After the cribbing is completed a temporary sinking head-frame is built either directly upon the crib or upon heavy beams resting upon it.

Sinking Head-frames.

Head-frames of very simple form are now used for sinking the largest shafts. That shown by Atlas plate, No. III, erected by the Lehigh Valley Coal company at the Prospect (Dorrance) large shaft, is a model of simplicity.

Standards of heavy square timber 10″×10″ form the skeleton of the frame, mortised and pinned together and stiffened by diagonal braces.

The frame is 17′×10′ at the base and 22 feet high to the sheave pillow-blocks. The sheave is eight feet in diameter and is set on the central line of the shaft, but not exactly in the middle.

The bucket is of boiler iron about three feet in diameter at the top and two feet six inches deep. It is suspended by a handle pivoted slightly below the center of gravity, and is locked in an upright position by a loose ring on the handle which slips over a pin on the rim of the bucket.

When a loaded bucket is raised the hanging chain shown by the illustration,* is hooked to the handle and when the rope is slacked by the engineer the bucket swings over the shute. The ring is then knocked up, freeing the bucket, which is then easily dumped. The rock passes down the shute to the dump-car standing on a track laid at one end of the shaft.

Rocks too large to be raised in the bucket are suspended by chains as shown by the illustration and removed by a tram-car running on a track laid on sills crossing the shaft inside the head-frame.

The shute is provided with a gate or door and will hold several bucket-fuls of rock.

The drawing also shows the engine, boiler, etc., which will be described in the following pages, and a steam pipe running down the shaft for ventilation.

Sinking Engines.

The majority of shafts and slopes have been sunk with old engines, but in the last few years engines especially designed for this work have been built, and are now often seen performing other work, after having been used to sink the main openings.

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* The chain had been made somewhat too long in the drawing.

A transferable sinking engine, built at the Hazleton shops, under the supervision of Mr. D. Clark, is shown on Atlas plate No. III. This engine has been used at several works of the Lehigh Valley Coal company, by Coxe Bros., and others, and is probably one of the best forms of sinking engine made. It is not as compact as engines built with the drum between the steam cylinders and the driving shaft, but its working parts are much more accessible.

The cylinders are twelve inches in diameter by twelve-inch stroke, geared at right angles; the valves are slide valves, placed on the side of the cylinder; the drum is geared to the driving shaft by a seven-to-one cog-gearing.

The drum is built of heavy cast-iron spiders, with wooden lagging four inches thick. The engine-bed is rather light, and made in several pieces, bolted together, so that it can readily transported. At the Dorrance shaft it is set on a crib of square timber.

Both the reversing and brake levers are within easy reach of the engineer, as shown by the engraving.

Drills and Explosives.

A comparatively small number of shafts have been sunk with steam or compressed-air drills. The old method of sinking by hand has still a firm hold in the anthracite regions, and many of the individual operators and companies consider it cheaper, and some more expeditious, than sinking by power drills.

It will not be necessary to describe the drills used, as nearly every style of percussion drill made in this country has been used in the anthracite fields for either sinking or tunnelling.

The objections to the use of power drills are:

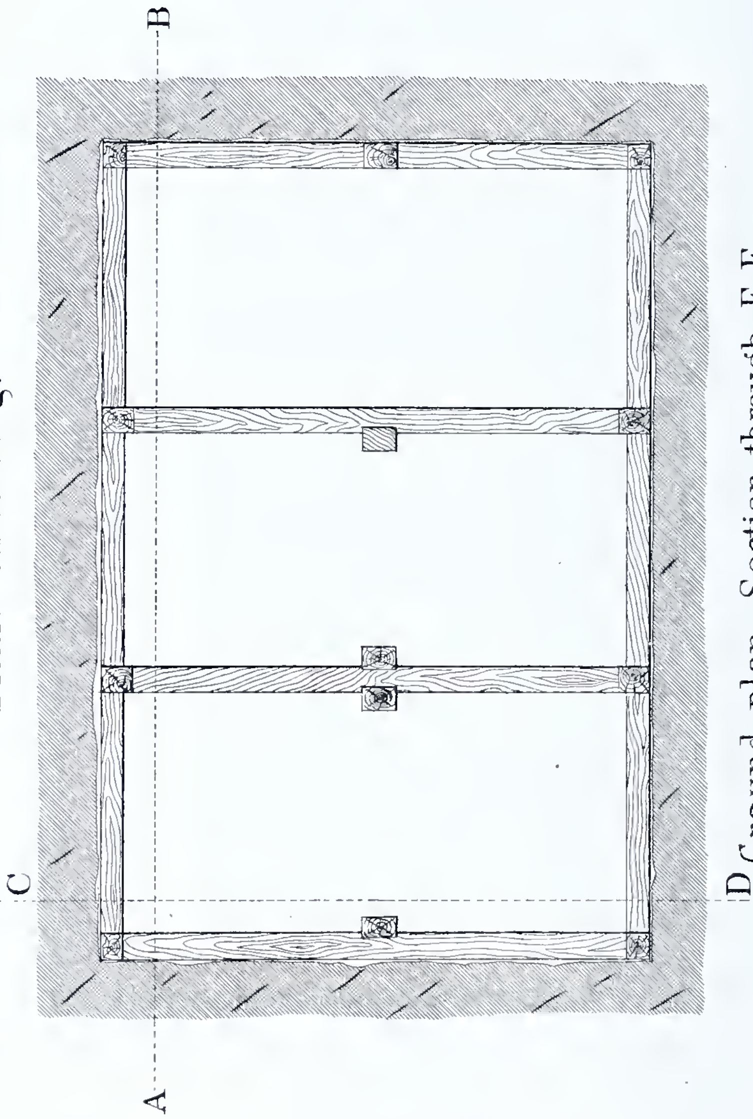
First. The cost of the drills, compressors and fittings.

Second. Time lost in removing the drills before blasting, and in placing them in position for drilling.

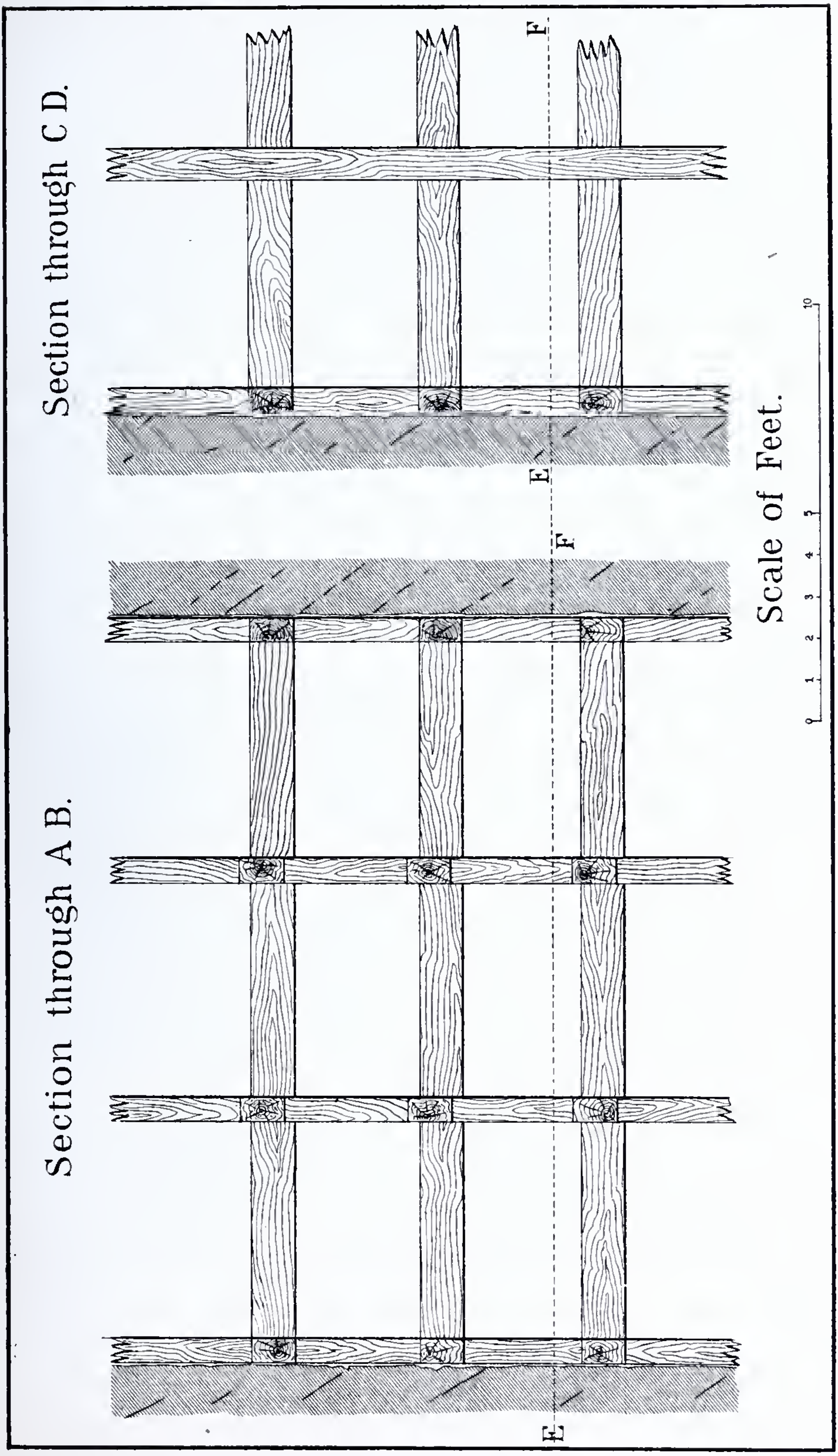
Third. Prejudices of the miners.

There is no doubt but that, under proper management, quicker and cheaper work can be done in hard rock with the steam drill or drill worked by compressed air than by hand,

Shaft Timbering.



Ground plan–Section through E F.


Section through C D.
Section through A B.
Scale of Feet.
0
1
2
3
4
5
10
E
F
E
F

but the difference is probably not so great as might at first be supposed.

In sinking large shafts the time lost in removing the drills might be lessened by working alternately on each end of the shaft, and removing the drills to the opposite end, under a shield of some kind, during the firing.

Nor is it necessary to describe here in detail the hand method of sinking, which is in every respect similar to ordinary quarry work—a gang of three men, two strikers and the drill-holder, working together.

The number of gangs employed at once is of course governed by the size of the shaft. In shafts of the size now ordinarily sunk, from three to five gangs work together at once, and when quick time is to be made, three shifts of eight hours each are worked per day; otherwise two shifts are worked.

The center, or that portion of the shaft immediately under the head-frame is kept in advance of other portions of the shaft to facilitate loading the bucket and to act as a sump for the collection of water.

The holes are drilled from three to six feet apart, depending upon the nature of the rock, its dip, cleavage planes, etc., and are now usually fired simultaneously in groups by an electric current.

"Rend rock" and other varieties of the higher explosives are used in hard rock, but in soft slate better results are obtained with ordinary black blasting powder.

The dip or cleavage of the rock often governs the position of the lowest point or sump, and in order to obtain the best results blasting it may sometimes be advisable to move the head-frame or to erect an auxiliary frame, in order to have the lowest point (*sump*) immediately beneath the sheave.

A plumb-bob is suspended from each corner of the shaft, either from the flooring or from a beam laid across the cribbing, to guide the miners in squaring the corners and sides.

After firing each blast a gang of laborers immediately descends to load the displaced rock while the miners resume drilling.

Timbering.

When the walls are self-sustaining the shaft is usually left untimbered, but sets of heavy single timbers (buntons) are placed from four to seven or eight feet (vertically) apart to divide the shaft into compartments. The cage-guides are fastened to these timbers by bolts with countersunk heads.

Under such conditions these timbers are simply fitted into notches or steps cut in the rock on each side of the shaft, and fastened tightly in place by wooden wedges.

When the sides of the shaft are not self-sustaining, complete timbering is necessary. Page plate No. 8 shows the heavy timbering and cribbing of the Hollenback shaft from a drawing by Mr. J. H. Harden. A light form of timbering for shallow shafts is shown by Page plates Nos. 6 and 7. The distance between each set of timbers is of course dependent upon the relative insecurity of the shaft walls,—from four to eight feet may be taken as an average.

Various forms of joints, mortices, &c., are used, but the form is not a matter of much importance, *provided* the strength of the timber is not greatly impaired by the wood removed, or a tendency to split created. Wedge-shaped joints and dovetailing are therefore bad forms.

Each set of timbers is firmly wedged in place, and at the same time lined up true, by wooden wedges driven in between it and the rock wall.

When the rock is very unsafe, lagging or sheathing planks are placed behind the timbers. They are either driven in and wedged in place (lagging) or, what is better, spiked to stringers running across between each set.

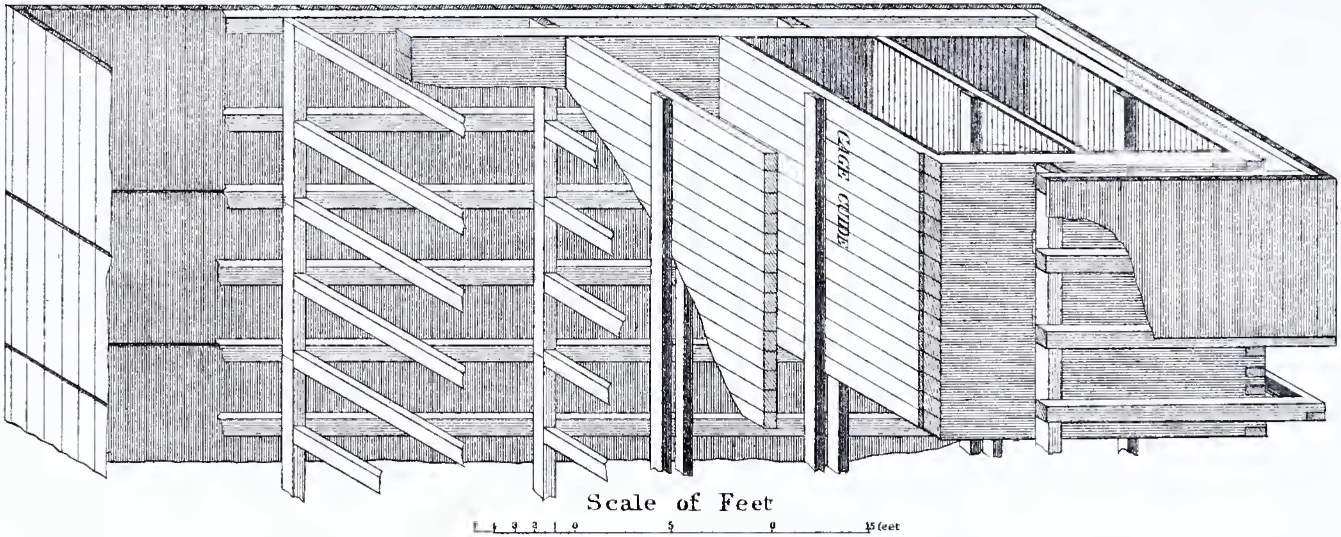
At the Hollenback shaft the permanent cribbing is made of 12″×12″ timbers measuring 45′ 4″×11′ 6″.

Outside the permanent cribbing and about one foot from it is a temporary cribbing built of 8″×10″ timbers. The backing or sheathing is of 3-inch plank and a clay filling was rammed between the two cribs to exclude the surface water. The total dimensions of the earth pit to base of cribbing is 51′ 2″×17′ 4″×31′ deep.

The plan here adopted of building a temporary crib-work as the pit is being sunk is (as has already been stated) fre-

METHOD OF CRIBBING THE HOLLENBACK SHAFT.

BY J. H. HARDEN, M. E.



Reproduced to illustrate report of H. M. Chance.

quently necessary where the earth has a tendency to run and fill the pit; and when there is a troublesome amount of water, no better and cheaper plan has yet been devised than the clay filling used in this instance. When the water is troublesome, and the soil or drift of any considerable depth, the filling should certainly be thicker than that used (12″) at this shaft.

Compartments.

All hoisting shafts are divided into compartments. At large collieries two compartments are usually used for raising coal, a third for the pumps or pump-rods and column-pipe, which is also often used as an upcast airway, and a fourth held in reserve for lowering timber and raising and lowering the miners, etc.

Two compartments are occasionally used in common as an airway and for the pumps, and sometimes two compartments are held in reserve for lowering timber ,etc., and for raising and lowering miners.

When the whole of the shaft is used as a down-cast, a very small compartment answers for the pump-rods and column pipe, and the practice is to divide the shaft into two large compartments for winding coal, men, and mine supplies, and one small compartment for the pumps, column pipe, etc.

A reserve hoisting plant is sometimes placed at the second opening and used exclusively for raising and lowering the miners and mine timber, etc., (as at the Exeter colliery.) this opening being at the same time used as an up-cast, the hoisting shaft being used as a down-cast, or as both up-cast and down-cast.

The size of the compartments depends upon the size of the cage, which is necessarily governed by the size of the mine car.

At the Hollenback shaft the compartments are 11′ 6″ × 7′; at the Exeter shaft (see Atlas Map II) the compartments are seven feet wide between the buntons.

The arrangement of buntons to carry the guides, brattice, pump rods and column pipe in an untimbered shaft is clearly shown by this Exeter shaft drawing. In this particular case they are placed at intervals of six feet from center to center and are simply let into the rock on each side a sufficient distance to give a good bearing.

The guides are in some cases simply bolted (by bolts with countersunk heads) to the buntons, but a better plan is to let them into gains cut on the bunton, as by this plan heavier guides may be used, and the bolts are subjected to less strain.

Atlas plate No. II also shows the arrangement of the buntons in the cribbing at the Exeter shaft, and the substantial masonry curbing, one side of which is used as a foundation for the head-frame.

A much lighter form of timbers is shown by Page plates 6 and 7 at a shaft at Mahanoy City, in which the main frame is made of 6″×12″ timber, the buntons the same size, the vertical braces or struts 6″×8″; the guides are 6″×8″ and are sunk into shallow gains cut in the buntons. This form of timbering is very common for shallow shafts. This shaft is 18 feet long by 12 feet wide and is divided into three compartments.

Practical Details—Sequence of Operations.

When the location and size of the projected opening have been determined, and staked out on the ground, an open pit is commenced in the loose earth and sunk as such to the bed rock. This pit is usually made from 4 to 8 feet, or more, larger than the size of the shaft, thus allowing from 2 to 4 feet on each side for masonry or timber cribbing. When the loose earth or soil is deep or when it is of a sandy character, or too soft to sustain itself, a light temporary crib is necessary (which is sheathed if the material is loose sand or quicksand) to sustain the walls until bed-rock is reached and the permanent stone-curbing or timber-cribbing built.

While the pit is being excavated preparations are made for erecting the hoisting engine and boilers. The winding (sinking) engine is best located opposite *one end* of the shaft and not opposite the side, for if the latter location is chosen it will interfere with the erection of the permanent plant and it will be impossible to shift the head-frame without throwing the rope out of line; again, with the engine opposite one end of the shaft two head-frames may be erected and both buckets may be operated from the drum without the use of deflection pulleys.

As the load to be lifted during the sinking operations is comparatively light, —from one to three thousand pounds,—the engine foundations are not massive. In some cases a timber crib ballasted with stone answers every purpose, but under ordinary circumstances a rough stone foundation of moderate depth or a combined stone and timber seat will cost but little more and give better results.

The engine foundations having been staked out, a suitable pit is dug, the foundations are laid, and the sinking engine and drum set up while the shaft-pit is being dug and cribbed.

When the pit is cribbed with timber, no stone curbing being used, the timbers are usually set "skin to skin"; the frames are securely mortised together and held in place by keys inserted in gains cut in each frame immediately opposite each other. For crib work curbing, square timber 10″×12″, 12″×12″, or 12″×14″, is commonly used, but larger timber has been used at a few shafts. Enough of the decomposed rock at the bottom of the pit is removed to expose a hard firm floor on which to set the cribbing.

The sides are braced by heavy buntons of square timber, which divide the shaft into compartments. These are sometimes not inserted when the cribbing is built, but the shaft is left clear of all such obstructions until finished.

In large shafts the buntons in the cribbing are placed only one or two feet apart, and in yielding ground are sometimes placed in juxtaposition, ("skin to skin").

When the pit is deep the loose earth is sometimes raised by a windlass; but the method usually adopted is either to cart it out through a cut, or to shovel it upon stagings, from which it may either be shovelled or wheeled away in barrows.

When a stone curbing is erected instead of a timber cribbing, the process is precisely the same; but it is necessary to place inside of this a timber framing (or buntons) to carry the keeps or wings, the cage guides, steam and column pipe, pump-rod guides, etc., etc., but these timber frames are then set some distance (usually from three to eight feet) apart. They are sometimes built into the masonry, but are frequently inserted afterwards.

When the pit has been completed, and the cribbing or curbing finished, the work of sinking through the rock may be commenced as soon as the sinking head-frame is built and the engine and boiler setting completed.

The head-frame is best built on heavy sills laid across the shaft and resting on the cribbing or curbing. These sills may be bolted to the cribbing to secure greater rigidity, and the frame itself is securely bolted down to the sills.

The head-frame being completed, the rope wound on the drum, the engine and boiler connections made, the work of sinking through rock begins.

When the rock is naturally soft and has a tendency to slip, or when a considerable thickness of partially decomposed rock is passed through before striking solid strata, the work of timbering must be carried downward *pari passu* with the sinking, but when the strata passed through are hard and self-sustaining, the opening is usually left untimbered until the sinking is completed ; and under such conditions the timbers are set at a considerable distance apart, their office being not to support the sides so much as to carry the guides, column-pipe, pump-rods, etc.

When the timbering must be carried down with the sinking, it is usually necessary to set the frames at short distances apart. Two methods of holding the frames in place are in use : 1. By wedging them fast by wooden wedges driven in between the frame and the rock. 2. By making the end or side pieces longer than the width or length of the shaft, and setting them in notches and wedging them in place.

When the timbers are to be wedged fast, they are held in place either by props or are suspended by chains or bolts from the set above, until firmly fastened.

In driving the wedges two objects are accomplished : 1. The timbers are securely fastened. 2. They are brought into a vertical line with the sets above, so that the shaft guides shall be perfectly perpendicular and straight, and the compartments of equal size throughout.

When the timbering is inserted after the completion of the sinking, the process may be reversed, the timbering being commenced at the bottom and finished at the top.

Water : When only a small amount of water is encountered, it is usually allowed to collect in a sump immediately beneath the head-frame, and is then raised in the bucket used for hoisting rock, but when numerous heavy feeders are encountered a steam (sinking) pump is necessary.

It will not be necessary to describe here any of the forms of pump used for this purpose, as almost all kinds of steam pumps have been so used. Rod-pumps have been used in the past at a few sinkings, but they are not used at present for this purpose.

Ventilation.

When the sinking is of moderate depth, a fire burning in one corner of the shaft will supply ample ventilation. If the powder smoke does not clear away in a few minutes a bundle of straw is taken down and burnt at one end of the shaft and a few pails of water thrown down at the other end.

When greater depth is reached, or if the shaft is of small cross-section, ventilation is accomplished either by a steam jet, by a fire drawing up into a large board pipe, or by a fan.

When a fan is used a portion of the shaft,—often one compartment,—is bratticed off, or a large board pipe is carried down to the bottom at one end of the shaft, and the fan is used either as a blower or as a suction fan. A suction fan will clear the smoke out more quickly and admit the light sooner than a blower, but by blowing good air is obtained *at the bottom* immediately after firing.

The fan sometimes takes power from the winding engine, (when the drum is provided with friction gear,) sometimes from a small engine that may be used for other purposes, and is sometimes turned by hand.

Fans used for this purpose are usually from two and a half to six feet in diameter.

The men are commonly raised and lowered on the bucket, but when the shaft is timbered as it is being sunk, ladders are usually provided.

Development.

When the shaft has reached and passed through the bed to be worked, a small excavation is made beneath the two hoisting compartments for a timber frame on which the carriage is to rest. On the dip side of the shaft a sump is excavated in the coal, and at the same time gangways are being driven to the right and left to open up the bed for mining.

But before much development work can be done it is necessary to erect the permanent head-frame, hoisting engines, boilers, fan, etc.

It is usual to proceed with the erection of the winding machinery, hoisting engine and boilers while the shaft is being sunk through the lower third or fourth of its depth, so that all will be in readiness for the work of development as soon as the shaft is finished.

The erection of the permanent head-frame may be completed before the removal of the temporary structure, so that there is no stoppage from this cause. This can be done at large shafts, by erecting the temporary head-frame over the portion of the shaft to be used as an upcast airway, or by making it so small that the permanent structure can be built around and over it.

When the permanent hoisting plant is completed, the work of development is prosecuted night and day until one of the gangways is holed through into the second opening, and until sufficient breasting area has been opened up to satisfy the operators.

As the work of development proceeds, or in many cases as soon as it is commenced, it becomes necessary to erect the permanent ventilating machinery—which is now usually a fan of the Guibal type—and to make preparations for the permanent pumping plant.

The hoisting compartments of some shafts are lined with boards from top to bottom; but if these are to be used together as a downcast airway, there must be free communication between the two compartments, and one side at least must therefore be left open.

Speed of Sinking.

The speed of sinking, including the time required for timbering and all stoppages, may be considered *to average* from 200 to 300 feet per year; and while many shafts have been sunk at a considerable faster rate, the above figures may be considered to represent the average progress of hand work.

The adoption of steam drills, or drills worked by compressed air, certainly increases the speed of sinking through hard rock; but in slate, shale, and soft sandstone, better time can probably be made by hand.

The fact that the use of steam drills is decidedly objectionable,—especially when the shaft is timbered as sinking progresses, or when the rock is of such a character that the heat of the steam causes it to "melt," soften, or slip,—and the fact that at anthracite collieries there is comparatively seldom any other use for air-compressing machinery, (more particularly at new collieries,) have together prevented the general introduction of power drills in shaft sinking.

Cost of Sinking.

In *very hard rock* the average cost of sinking shafts of average cross-section is from five to eight dollars per cubic yard for shafts from 400 to 800 feet deep. The cost increases rapidly with the depth, and for shafts more than 800 feet deep the cost per cubic yard may approach ten dollars.

But in softer strata, especially shale and soft sandstone, shafts from five to eight hundred feet in depth have been sunk for much less. The average cost per cubic yard in such material ranges from two to five dollars per cubic yard.

These figures apply only to sinking by hand; in hard rock power drills probably give better results, but I have not been able to demonstrate this with authentic figures.

Taking an average alternative of hard and soft rocks such as is passed through by the shafts in the Wilkes-Barre basin, we may consider from three and a half to four dollars per cubic yard a fair average for shafts of ordinary depth and size.

Long-Hole Diamond Drill Process.

The long-hole process of sinking by the Diamond drill was successfully used at the Pottsville (Norwegian) deep shafts, a minute description of which may be found in a paper from the pen of Mr. Eckley B. Coxe in Vol. I of the Transactions of the American Institute of Mining Engineers.

The east shaft, used as the main hoisting shaft, is nearly 1,600 feet deep and is 11′ 6″×15′ 8″ in cross-section, inside. The size of rock cut was 13′ 10″×16′. The west shaft is 25′ 8″×13′ 10″, or inside the timbers 23′×11′ 6″.

Four drills were used*, each boring a hole an inch and three quarters diameter and from two to three hundred feet deep. In the east or hoisting shaft twenty-five holes were bored about three feet and three inches apart in one direction and about four feet apart in the other direction.

When completed the holes were filled up with sand and the drills removed to the west shaft. The miners then commenced blasting by removing three or four feet of sand from the holes and firing them in groups. This process was continued until the bottom of the holes was reached, when the drills were again changed, and the work of drilling recommenced A central group of holes was always fired first and the outside rows afterwards. The outside rows of holes squared the shaft up very nicely so that but little trimming was needed, although many of the holes were not exactly perpendicular, and some were not straight.

An attempt was made in 1875 to sink the Harris navigation deep pits (in South Wales) by the Diamond drill, an account of which may be found in the proceedings of The

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* Seven were on the ground.

Institute of Civil Engineers, Vol. LXIV, Session of 1880–81, Part II, but after sinking the shaft from 175 yards to a depth of 244 yards this process was abandoned.

In sinking large shafts there seems to be little doubt but that the long-hole process is expeditious, but the financial results do not seem to be satisfactory.

When, as at the Ellengowan shaft, the holes can all be bored at one operation from the surface to the coal bed to be developed, the long-hole process will probably give more satisfactory financial results—but this is possible only at shafts of very moderate depth.

Kind–Chaudron Process.

This, which is nothing more than the oil well method of drilling on a gigantic scale, has not been used in the anthracite regions, nor to the author's knowledge in America.

An illustrated description of this method with statistical tables may be found in Vol. V, of the transactions of the American Institute of Mining Engineers, page 117.

CHAPTER V.

Slope Sinking and Timbering.

In the anthracite regions the term "slope" is applied to an inclined passage driven down on the bed (through coal), and through which coal is raised and mine supplies lowered. When the coal is lowered down an inclined passage, the loaded cars bringing up or assisting in raising the empty cars, the passage is usually termed a "plane." When an inclined passage is driven partly or wholly in rock, shale, or slate it is termed a "rock-slope." Such a slope is seen in Fig. 30

The following summary description of slope-sinking and timbering will, therefore, be understood to apply as well to planes. The method of sinking rock-slopes is of course somewhat different.

When the location of a projected slope has been determined, the first procedure is the erection of a temporary sinking plant. This usually consists of an old engine and drum placed upon a temporary timber foundation or a rude crib-work loaded with stones. A rough frame house is erected over the engine, and one or two tubular boilers or two to four plain cylinder boilers supply the steam.

For a short distance—varying with the nature of the ground but usually ranging from twelve to eighteen feet in depth—the sinking is carried down as an open cut and the earth and rock are thrown out by hand.

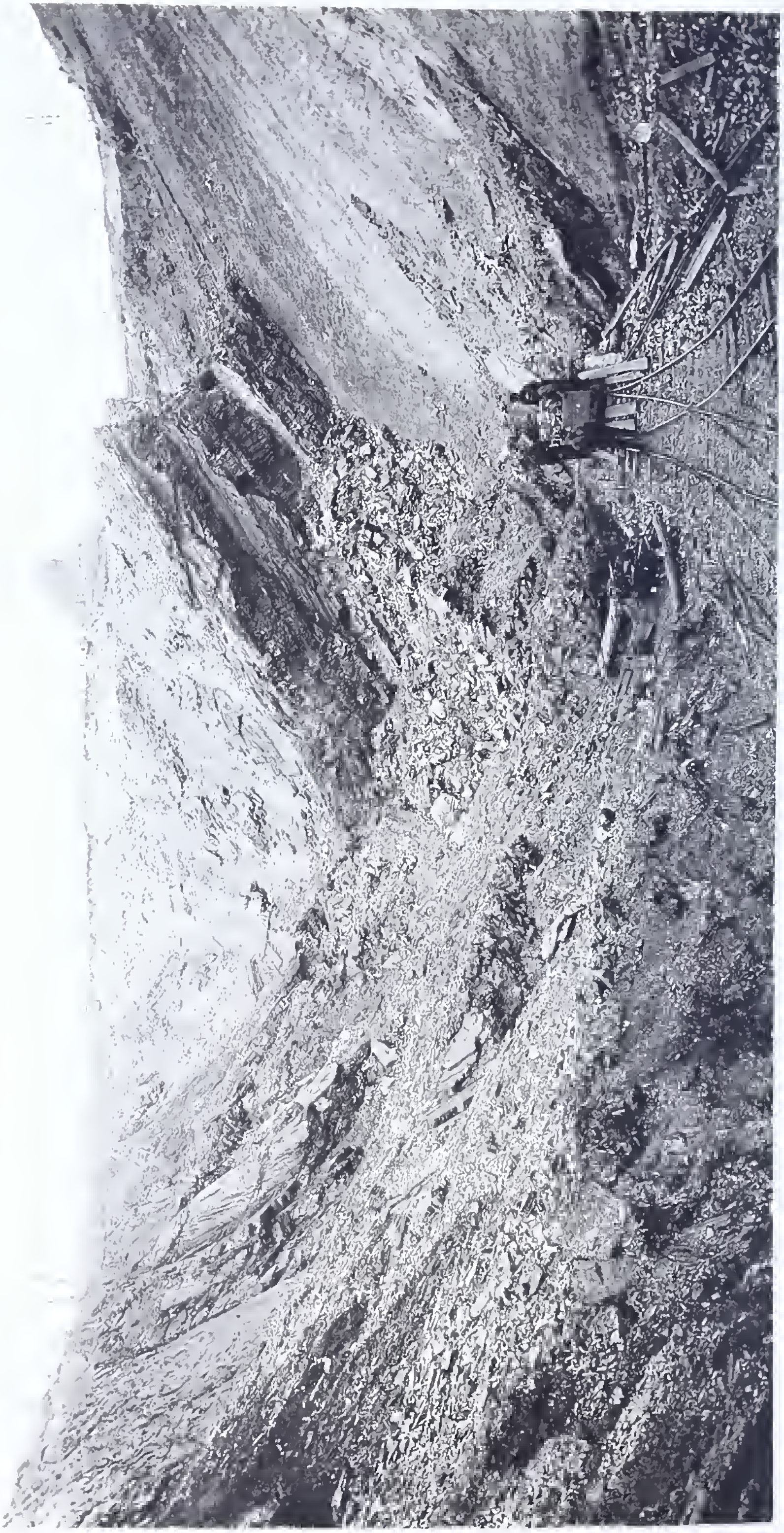
As soon as the roof becomes sufficiently strong the miners begin to work in under it, and the work of timbering is commenced. At the same time a single or double track is carried down and the coal and other material is raised in a car or self-dumping skip.

When a skip is used, the track is continued up on a trestle some distance above the mouth of the shaft, and a head-sheave placed in proper position to draw the skip up the required distance to dump the material into a dump-car or pocket beneath the trestling.

The width of the sinking depends upon the width of the cars and the number of compartments necessary. A common arrangement is to divide the opening into three compartments, two of large size for raising coal and a third of somewhat smaller size for the pump-rod, column-pipe, and for use as a traveling-way and this is also used as an air-way during the sinking.

The practice sometimes adopted of having but one hoisting compartment laid with three rails with a turn-out at the middle of the hoist, while cheaper at first, is of doubtful economy, as rope breakage on one side or falling coal from the upper wagon, is likely to demolish the other, and on planes and slopes so constructed, collisions at the turn-out

2^nd GEOL. SURVEY OF PA. REPORT A C. PAGE PLATE NO. 52.



STRIPPING AT HOLLYWOOD COLLIERY NO. 1,-- LOOKING EAST. (PICTURE REVERSED.)

ARTOTYPE, E. BIERSTADT, N Y

are not infrequent. The same objections apply to slopes with a single track and a passing turn-out at the middle.

It will hardly be necessary to refer here to slopes with a single track throughout. They are used only when the output is small. Slopes designed for three compartments as above described are usually driven from sixteen to twenty-two feet wide.

When the dip does not exceed thirty-five or forty degrees, the height of the slope is made about the same as that of the gangways, usually about seven feet in the clear from the top of the rail to the under side of the collars, measured not in a vertical line, but at right angles to the dip. On steeper dips the prevalent custom of raising the cars on slope cages (to prevent spilling the coal) necessitates a much greater height; but when the car is raised in gunboats,* the ordinary height suffices.

In slope sinking the operation of getting out the coal is similar to gangway driving, and is done with similar tools; but when in order to straighten the slope, or for other reasons, the driving is done in rock, rock drilling tools and explosives are used.

The bottom of the slope is kept in advance of the top or *vice versa* by putting in holes nearly vertical or inclined slightly down the pitch, and one corner is kept in advance of the main face to collect the water and give a loose end in blasting.

After the bottom has been underholed by a series of shots, the top is blown down as in gangway work; but this is not the universal practice, as the top is sometimes kept in advance of the bottom, and the lower benches blown up afterwards.

The timbering and track laying are carried down *pari passu* with the sinking, and kept within a few feet of the working face.

Slope timbering.

This is similar in many respects to gangway timbering, but the joints are cut with more care, and square timber is used much more largely for slopes than for gangways. It also often differs from gangway timbering in many other particulars, as shown by Page-plate No. 10, and by Figs. 10 to 15.

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* **Monitors or skips.**

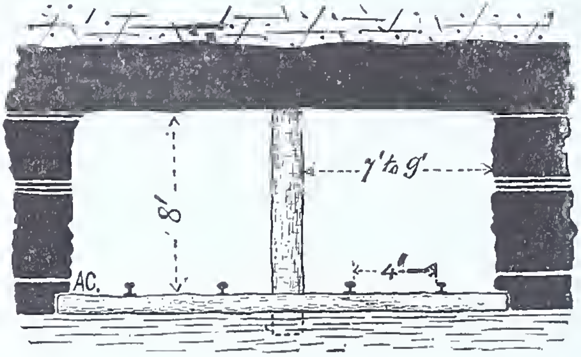


Fig. 10.

The above illustration (Fig. 10) shows a slope with no other timbering but a center-prop. On steep dips a heavy foot-sill is let into the rib on each side to prevent the road bed from slipping down the pitch.

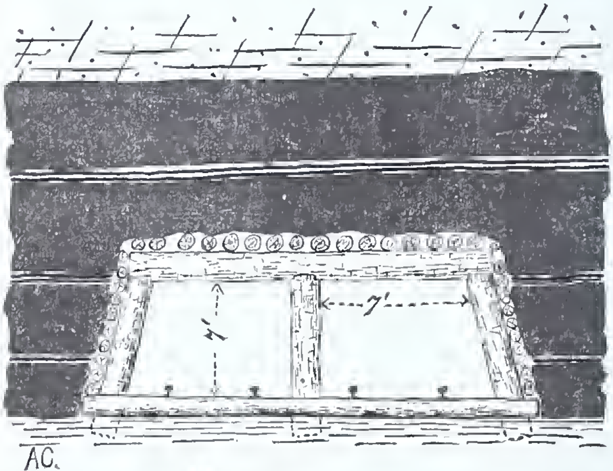
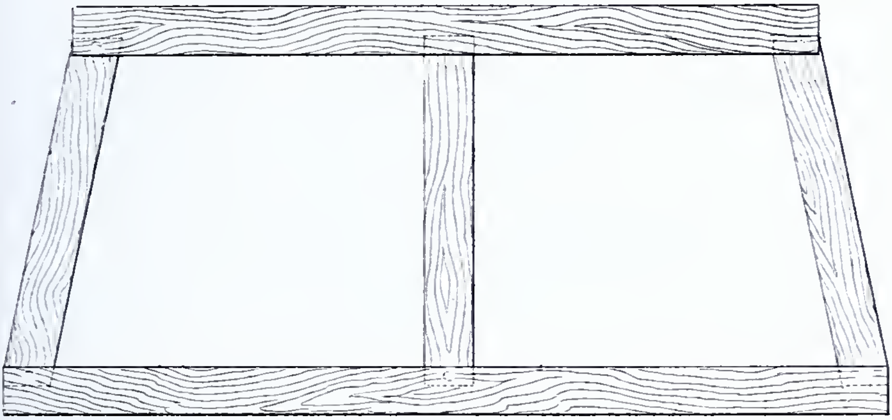


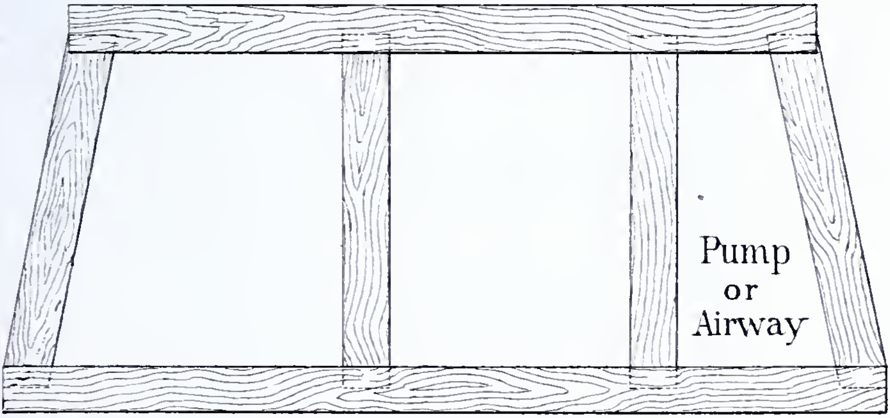
Fig. 11.

When complete timbering is necessary, the form shown by Fig. 11 is frequently used. The legs and collars and center-prop are similar in every respect to heavy double track gangway timbering, and they are surrounded by lagging similarly placed. (The timbers are too massive in Fig. 11.)

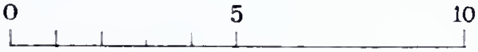
Slope Timbering



Square timber–Two Compartments



Square timber–Three Compartments



Scale 1" — 5'

More substantial forms of timbering are shown by Page-plate No. 10, in which square timber is used, with a heavy foot sill of square timber of stuff similar to the legs and collars. The compartments are separated by center-props of heavy square timber.

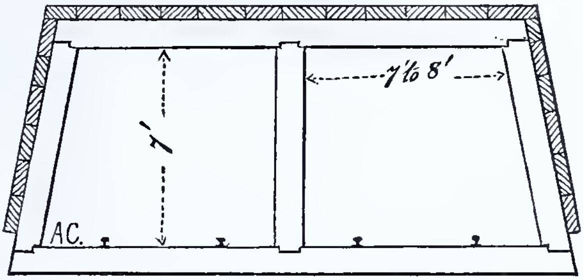


Fig. 12.

The timbering shown by Fig. 12 is for a two-track slope, gauge 4 feet, with plank lagging.

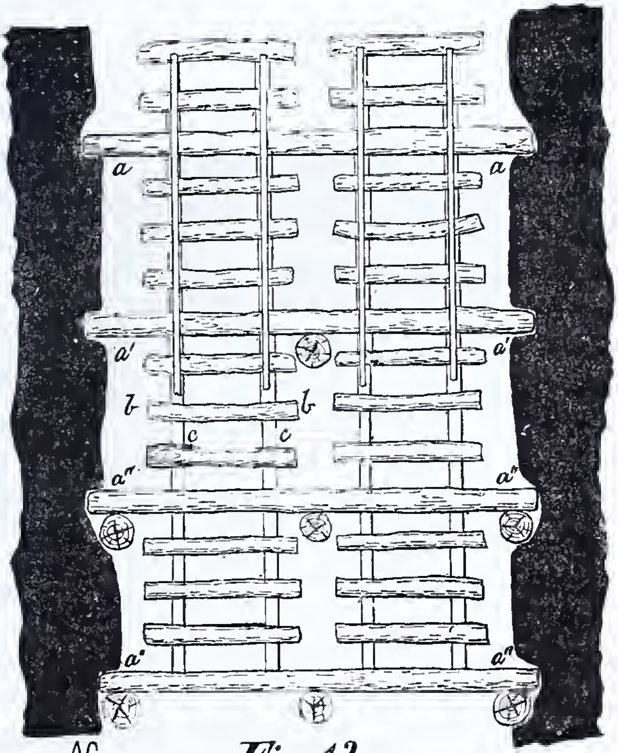


Fig. 13.

The distance between each two sets of timber is of course governed by the nature of the roof,—the usual distance is from three to eight feet, but in some cases (see Fig. 15) it is necessary to place them almost in contact with each other.

A ground plan of the arrangement of the foot-sills, legs, center-props and road-bed is shown by Fig. 13.

The foot-sills are shown resting against the legs and center-prop, in the two lower sections; the next segment shows the arrangement of the road-bed and the sills when only a center-prop is used and the upper sections show the road bed in a slope without any timbering whatever; *a*, *a'*, are main foot sills; *b*, *b*, are sleepers; *c*, *c*, *c*, are braces to keep the sleepers in place.

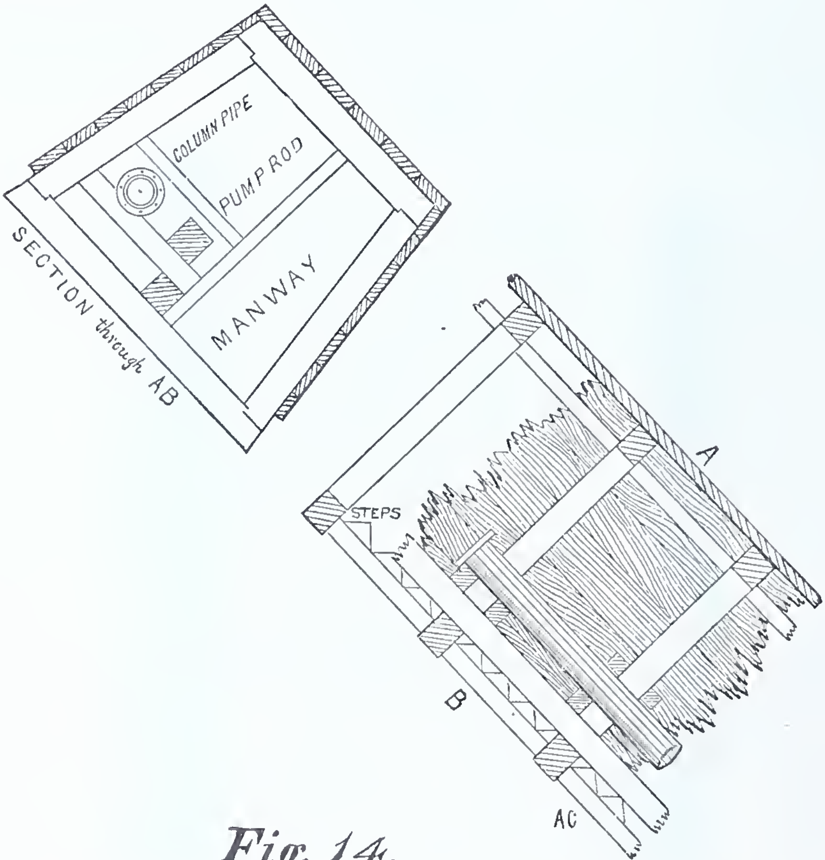


Fig. 14.

A transverse and longitudinal section of a slope arranged for use as a man-way and pump-way are shown by Fig. 14.

The column pipe rests on short beams that are also used to carry the guides, etc., of the pump-rods. The man-way is provided with a flight of steps, a convenience not always found at anthracite collieries.

The lagging used on slopes is generally of better material than gangway lagging, and greater care is taken in driving it to get the timbers into line. On well built slopes, timbered with square timber, two or three inch plank lagging is used.

Ventilation.

When a slope is timbered for three compartments the small pump—or man-way, is usually bratticed off and used as an upcast, a small fan being erected near the mouth of the slope and connected with this by a frame air-way, but when the slope is driven to open up a large body of new coal and an independent air-way is necessary, ventilation is maintained by cross-headings connecting the airway and slope.

Air-ways driven through the solid, to supply ventilation to slope collieries, are always at least ten yards distant from the slope, and in good mining practice this distance is increased to fifty or one hundred feet.

Ventilation during the sinking, especially in small single slopes, is frequently accomplished by means of board pipes, as described in the chapter on ventilation.

Drainage.

While sinking, the slope is usually kept free from water by a small steam pump until the depth is reached at which the first lift is to be opened. A sump is then excavated below the level of the gangway and in thick seams on one side of it, and the permanent pumping machinery is erected.

The pumping plant is occasionally built and the rods lengthened at the same time with the sinking, but the amount of water coming into a new slope is seldom sufficient to warrant the employment of this plan.

The *cost of sinking* depends:

1st. Upon the dip, thickness and character of the seam, 2d. Upon the size of the slope, and

3d. Upon the character of the roof,—whether of coal, slate or rock.

Under ordinary conditions a slope seven feet high inside the timbering, with two hoisting compartments and a pumpway, can be sunk and timbered for a total cost of from thirty to fifty dollars per lineal yard.

When the roof is very bad the cost will be increased by the necessity of using very heavy timbers and setting them close together. With a fair roof the timbers are commonly set from four to eight feet apart; when the roof is very good only a single row of center-props may be used, and in some slopes the roof is of such an excellent character that no timbering is required.

Fig. 15 shows the timbering of a slope near Mahanoy City where the ribs and roof were so soft and yielding that large round timber was used and set "skin-to-skin." The great height of this slope is occasioned by the use of a slope carriage.

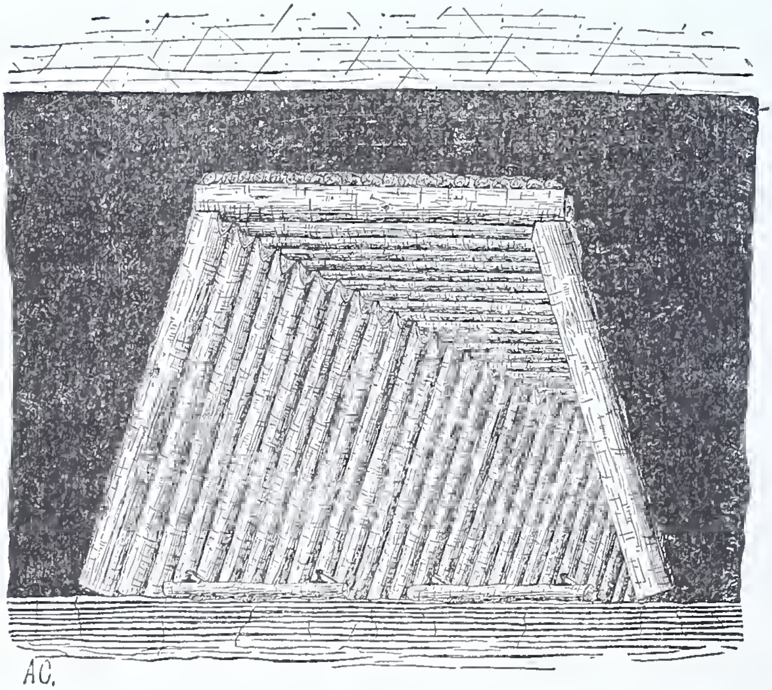


Fig. 15.

When no foot-sill is used and the legs and props are slipped down into pot-holes at the sides and along the middle line of the slope, a long sill is laid across the floor and is held in place by resting against the lower portion of the legs and center-props. To this sill the rails are spiked, as well as to short sleepers between the timbers. See Figs. 11 and 13.

When only a center-prop is used in timbering, or when no timbering is used at all, long sills are laid across the floor at intervals of from six to twelve feet, (varying with the dip,) and the rails fastened to them as in the former case, as shown by Figs. 10 and 13.

The sleepers are wedged in place by short struts, from ten to fifteen inches long, marked *c* in Fig. 13.

These precautions, necessary on steep dips to prevent the road-bed from sliding down the pitch, are unnecessary on more gentle dips, where the road-bed may be built precisely like a gangway road without bracing the sleepers.

Long experience has taught the miners of coal in the anthracite regions that at slopes on which a large amount of coal is to be raised, it is advisable to have two single and entirely independent tracks throughout.

Several different designs for the arrangement of tracks at lifts and at the bottom are, shown by Figs. 35 to 39 inclusive, and described in the chapter on underground (inside) slopes.

Some of the objections to three-rail slopes have already been noted ; the objections to a single track with a turnout in the middle are still greater ; and even the practice of uniting the two tracks at the top and bottom is objectionable.

The best plans are those that do away entirely with switches and turnouts *on the inclined portion* of the slope, and have two entirely independent tracks throughout, the switching being done at the top and bottom, at which points the cars are to great extent handled by gravity.

Square timber less than 12″×12″ is seldom used in slope timbering, and when round timber is used the legs and collars are made of as large stuff as possible, especially if there is no central prop.

The expense incurred and time lost in retimbering a slope being much greater than in retimbering a gangway, it is of considerable importance to timber with material that will not soon need replacing, especially when square-framed timber is used with a foot-sill, for it is then often necessary to tear up the track and rebuild the road-bed when the timbering is renewed.

Slopes timbered without a center-prop usually need heavy collars of from 16″ to 22″ round timber, but as this very heavy stuff is not readily obtained, 12″ to 15″ stuff is often used where 18″ timber is really needed.

CHAPTER VI.

Gangway and Tunnel Driving.

As Anthracite coal seams are usually inclined at a considerable angle, and as the beds now worked are the largest and best of the series, gangways can be driven of almost any reasonable height. In mining bituminous seams the height of the gangways is usually governed by the thickness of the seam.

Gangways are usually driven of sufficient height to give about seven feet in the clear from the rail to the under side of the collar.

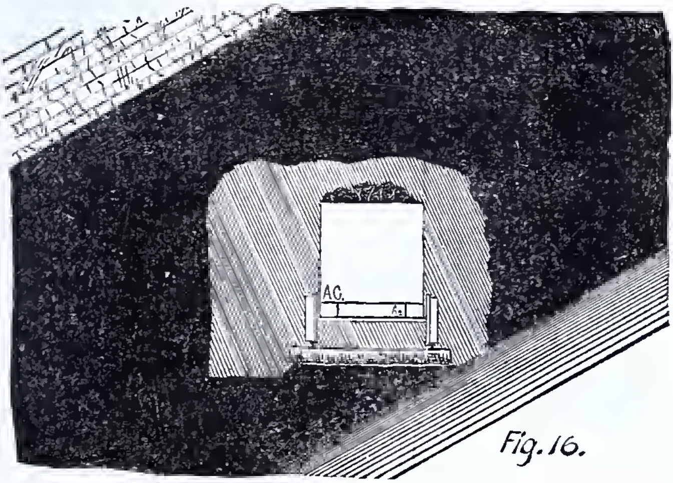


Fig. 16.

When no timbering is required, a height of from 7′ 6″ to 8 feet is sufficient, allowing from 6 to 9 inches for the height of the sill and rail. Such a gangway is commonly driven from 9 to 10 feet wide, with a ditch or gutter excavated on one side of the roadbed.

Grade: A fall of from 6 inches to one foot in a hundred feet is sufficient to carry the water, even if the ditch is not kept perfectly clean. A fall of 1 foot in 125 to 150 feet is found to give the best results in the haulage of loaded and empty cars, and the adoption of such a gradient, say three fourths of a foot in one hundred feet, will probably give the best results. A gradient of one foot in two hundred is largely used, but this, although sufficient to carry the water, probably does not give as good results for a road gradient as a somewhat greater fall.

In the Wilkes-Barre region the gaseous condition of all the deep mines necessitates larger gangways than those commonly in use in the Schuylkill and Lehigh regions; but this increased size is rarely obtained by increasing the height,—usually by driving them of greater breadth, and in these collieries we consequently find gangways driven twelve and fourteen feet broad.

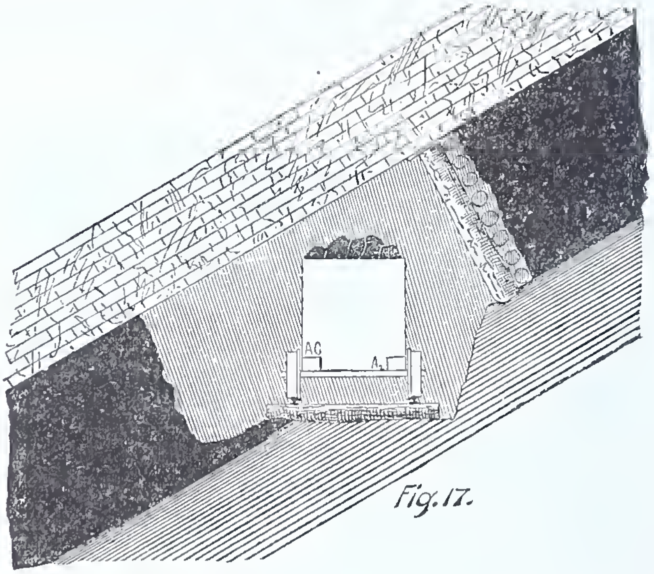


Fig. 17.

When the seam is thin with a good roof, or when the gangway is driven at the top of the bed, a single prop may be the only timbering needed. Such an entry is shown by Fig. 17, in which the prop is erected to prevent the loose coal from falling and blocking the road-bed. It also gives some support to the roof.

A similar form of timbering is shown by Fig. 18, termed the "Post and Bar." It is open to many objections, which are so self-evident that it is unnecessary to state them in detail.

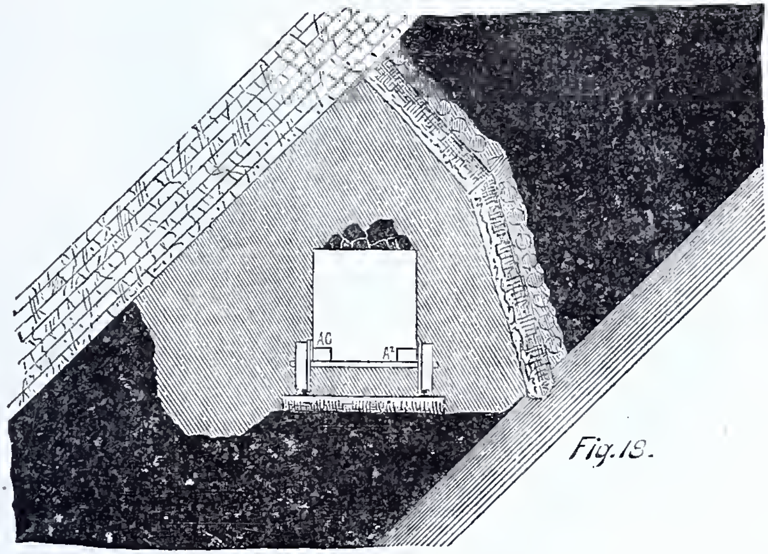


Fig. 18.

In all of these forms, and in others described further on, it will be noticed that there is a clear height of seven feet between the rail and the top of the gangway.

Double track gangways (or "turnouts" on single track gangways) are usually driven from sixteen to twenty feet wide, and require very heavy timber, especially if timbered without a center-prop.

Center-props: But few mine superintendents favor the use of center-props on turnouts, and even on double track gangways, and especially on turnouts at the bottom or at a slope lift, they are very objectionable. A large percentage of the accidents to mules, and a considerable number of accidents to drivers and miners, are chargeable to the use of center-props. Besides this risk of being crushed between the props and cars, the center props are objectionable on account of the space they occupy, and because they are in the way of a free passage for the mules from one track to the other.

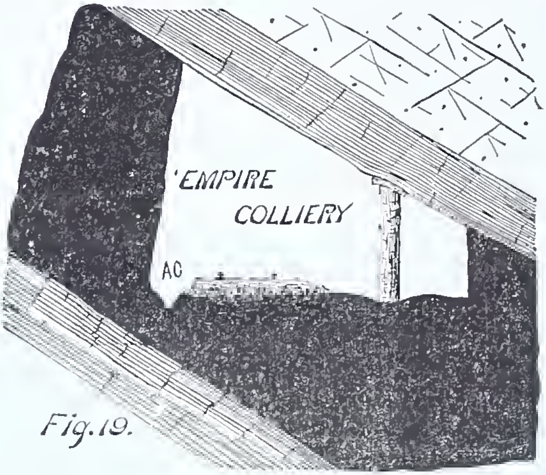


Fig. 19.

Gangways driven in comparatively thin seams, with a fairly good roof, are not infrequently timbered as shown by Fig. 19, at the Empire Colliery, near Wilkes-Barre, in which a single prop is used to support the roof, the coal above the gangway being self-sustaining.

When the roof is not good, and *complete timbering* is necessary, the form shown by Figs. 20, 21, and 22 is commonly used. In fact, probably one half of the timbered gangways in the anthracite regions are timbered with this style of timber. Timbering of the same style, but smaller, is commonly used in small cross-headings and airways.

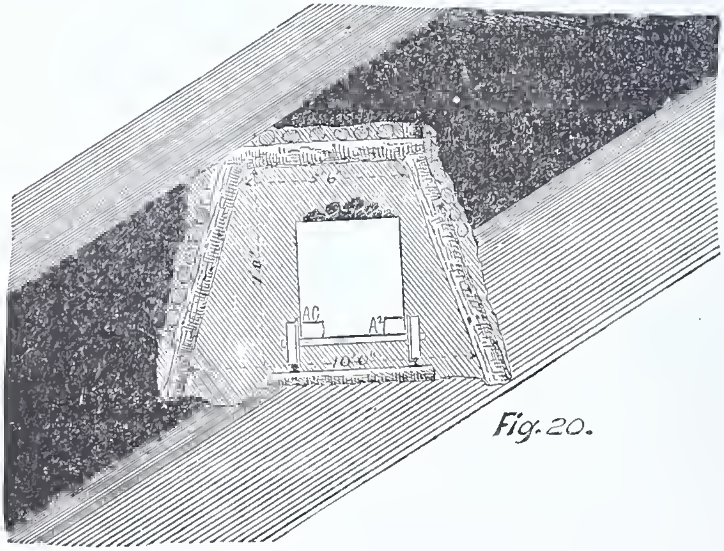


Fig. 20.

When the roof is not very bad, and the coal seam is moderately hard, the small sized timber shown by Fig. 20 is used, in which the legs are about nine feet long, the collar from seven to eight feet long, and both the legs and collar from eight to ten inches in diameter.

When the roof is very bad, or when the seam is soft and the coal has a tendency to "run," large timber as shown by Figs. 21 and 22 is used.

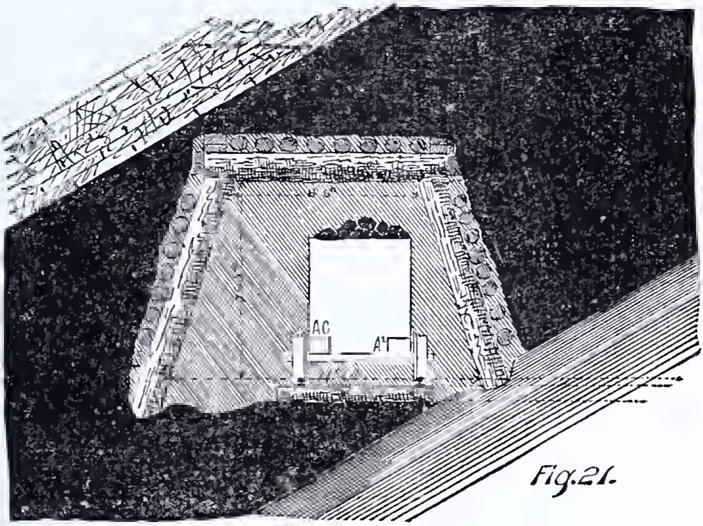


Fig. 21.

The legs and collar are of about equal length,—from eight and a half to nine feet, and from twelve to sixteen or eighteen inches in diameter. In Fig. 21 the right-hand leg is made shorter than the other, because it stands upon the floor of the bed.

The wooden wedges by which the collar is forced firmly down upon the legs, are shown by Figs. 20 and 21, and there are also wedges driven in at both ends of the collar to fasten it more securely in place.

The foot of each leg is stepped down into a pot-shaped hole to prevent it from being moved by lateral pressure exerted by the rib on either side.

Fig. 22 shows a longitudinal section taken along a gangway timbered as shown by Fig. 21.

In this drawing (which together with Figs. 16, 17, 18, 20, and 21 have been reproduced from Mr. J. Price Wetherill's

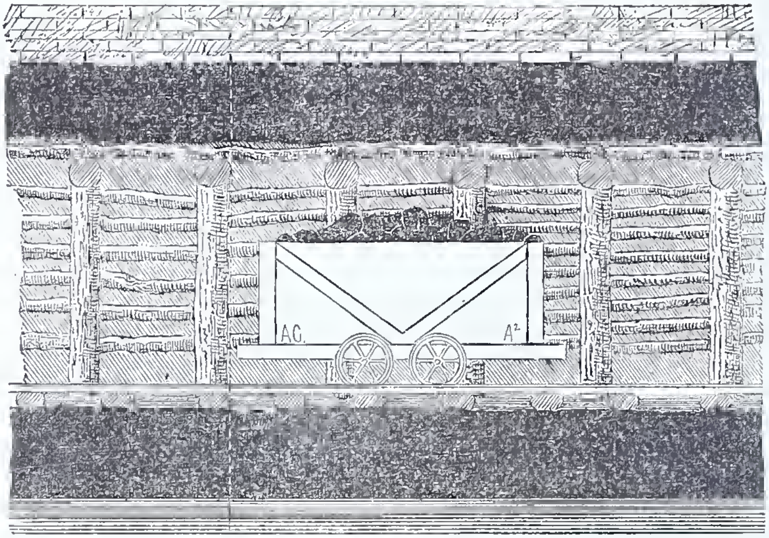


Fig. 22.

designs) the legs are set about four feet and a half apart from center to center. This distance of course varies with the character of the roof, etc., but when this method of timbering with complete lagging is required, the frames are rarely set more than five or six feet apart and often much closer, so that this distance of four and a half feet may be considered the average for this style of timbering.

When the coal is hard and firm, but the roof needs support, one leg may be saved by the plan shown by Fig. 23, in which one end of the collar is fitted into a recess excavated in the coal near the roof.

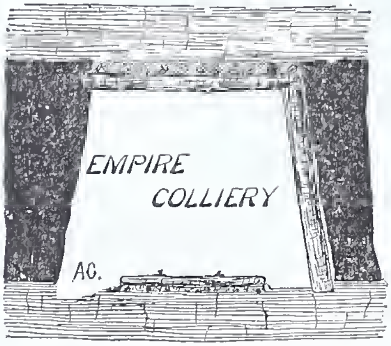


Fig. 23.

In seams of moderate thickness inclined at a considerable angle, it is necessary to take up a portion of the floor ; and if this bottom rock is hard and firm, a reduction in the cost of driving is effected by timbering with one short leg as shown in Fig. 24, which represents a plan in use at the Prospect Colliery of the Lehigh Valley Coal company.

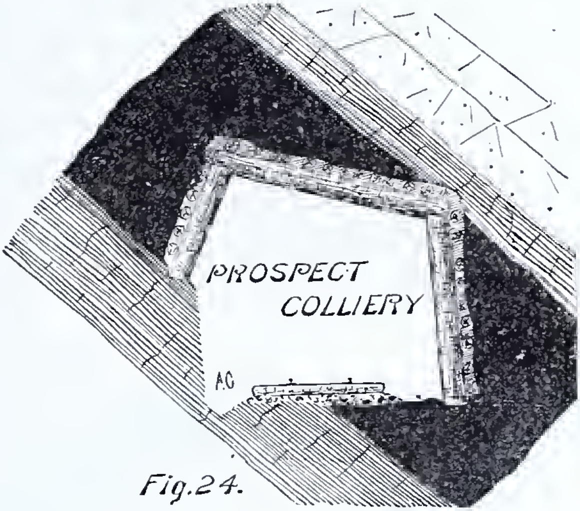


Fig. 24.

A form of timbering in use at the Hollenback Colliery of the Lehigh and Wilkes-Barre Coal company is shown by Fig. 25.

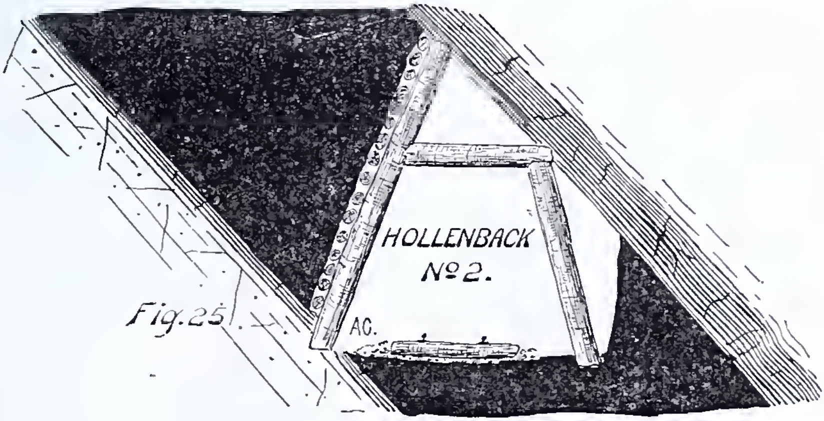


Fig. 25.

It is only necessary when the top coal is so soft that it comes down in driving the gangway, leaving a high face of unsupported coal on the upper side of the gangway. The long prop supporting this coal receives support near the middle from the collar, and the collar and leg in turn furnish some support to the roof.

This form and the following shown by Fig. 26 are applicable only under exceptional conditions; and to meet the various requirements of gangway timbering under ever-varying conditions, many other forms of timbering are and have been used, but to illustrate all of these would multiply the illustrations of this report beyond reasonable limits.

It will be sufficient to indicate the fact that nearly all forms of gangway timbering in use in the anthracite region are modifications of the regular style shown by Figs. 20 and 21, made by shortening or lengthening one or both legs, or by inclining them at different angles according to the dip of the bed, etc., etc.

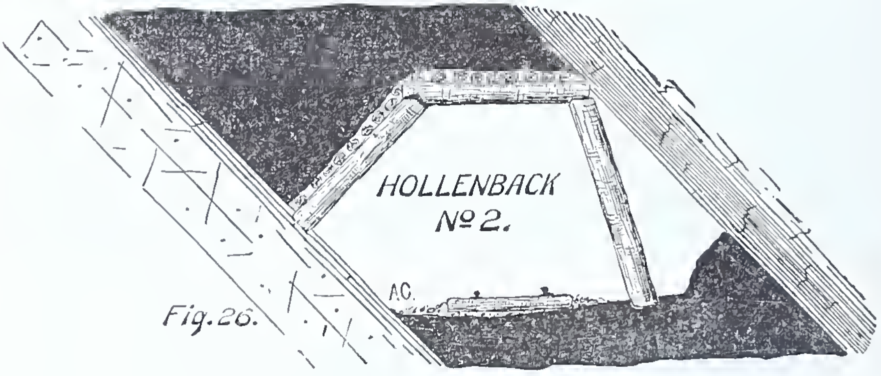


Fig. 26.

Many engineers are of the opinion that the legs, as nearly as is practicable, should be set at right angles to the dip, and propping is always so set, but in gangway timbering this can seldom be done, and probably an equal number of mining engineers believe that the best form of timbering is the ordinary upright form of Fig. 21.

Gangway Driving.

In opening up new workings at a moderate depth beneath the surface or above water level, ventilation is often affected by a brattice carried along the lower side of the gangway as shown by Fig. 51, and Page plate No. 38; but the plan usually adopted is to drive an airway parallel to the gangway, and connect them at regular intervals by cross-headings, as shown by Figs. 55 and 56.

The thickness of the pillar left between the airway and gangway is to some extent governed by the thickness of the seam. In the flat workings of the Wilkes-Barre district a pillar of from five to eight yards is usually left between the airway and gangway, and cross-headings connecting them are driven every ten or fifteen yards, or more.

When the coal bed dips at a sharp angle, the airway is commonly driven above the gangway as shown by the drawings on Atlas plates XXI, XXII, and XXIII.

When the roof is good, requiring no timbering, and the coal gives off such a small quantity of gas that separate airways are unnecessary, ventilation of the gangway heading is frequently accomplished by means of broad pipes, as described in detail in the chapter on ventilation. Board pipes and a small fan are also used to keep the heading free from gas in very gaseous mines, but for this purpose a brattice carried well up to the face is more efficient.

Single track gangways from eleven to fourteen feet spread at the base, and of sufficient height for a nine foot leg, and of a width at the top to admit a seven and a half to a nine foot collar, can be driven and timbered in hard coal at a rate of about two and a quarter feet per eight hour shift, or without timbering at a rate of nearly three feet per day. In the softer coal of the Wilkes-Barre and Schuylkill regions an average of four feet per eight hour shift can be made in driving untimbered, and about three feet for timbered gangways.

Gangway driving is usually done by contract at so much per yard, and the miner taking the contract hires and pays two laborers and bears all expense for oil, powder, tools, wick, etc.

The following table shows the range of these contract prices in the southern district of Carbon and Luzerne counties during the years 1875 to 1880 inclusive:

These contracts are made on two bases, 1st. the company owns the coal obtained in driving, or 2nd. the miner owns

Year.	Remarks.	Timbered Gangway.	Untimbered Gangway.
1875, . . .	. . . . .	$6.12	$5.35
1876, . . .	Lehigh District.	$5.20 to 6.48	$4.55 to 5.68
1877, . . .	do. do.	4.02 to 5.20	3 50 to 4.55
1878, . . .	do. do.	4.34 to 5.50	3.78 to 4.81
1879, . . .	do. do.	4.32 to 6.12	3.78 to 5.35
1880, . . .	do. do.	5.20 to 6.12	4.54 to 5.34

the coal and the company pays the contractor so much per car for all the coal (at miner's rates); these two being known by the terms "company's coal" and "miner's coal." The above table is on the basis of "miner's coal."

The contract price of course varies widely with the character of the coal, its thickness, and the size of the gangway; thus in 1879 in the Hazleton District the variation was $1.80 per yard as given in the table, and many cases might be given in which the variation is much greater.

From six to eight pounds of powder per yard of gangway eight by twelve feet is a fair average for hard coal, but in soft coal one half of this amount will exceed the average quantity required. The contractor's incidental expenses will average about one third the cost of his powder, so that his total expenses for supplies may be placed (for hard coal) at about 90 to 92 cents per yard, pay of laborers from $2.50 to $4.00 per yard, averaging $3.60, making his total average expenses in hard coal $4.50 per yard.

The profits or net earnings of miners working on gangway contracts must always vary widely with the individuals. Where one man will clear ninety or a hundred dollars a month another will make only forty or fifty. The average compensation of men doing this class of work is probably at present from sixty to eighty dollars a month.

The method of driving is similar in many respects to breast working, but some miners prefer blowing out the top first and afterward putting in four to six shots to square up the sides and blow up the bottom. The character and position of partings of slate or bone, and the direction of dip, and of the main cleavage planes always determine the plan to be employed. In some cases it is advantageous to to put the bearing-in shots on one side and then having a free end to blow out the other side by shots located near the top and bottom corner; sometimes to fire a group of two or three central holes, and then to blow out the top, the sides and the bottom by a series of peripheral holes.

The holes are usually drilled four feet, more or less, deep, but in loose coal, holes of a greater depth can be advantageously employed.

In thick seams the gangways are generally driven near the floor of the bed, but this location is not always adhered to in steep-pitching seams. On moderate dips this location of the gangway increases the pitch of the shute and facilitates loading, and on steep dips it allows room for the airway near the roof of the bed, but increases the pitch of the shute.

When the floor is very uneven the gangway is sometimes driven as nearly as possible in a straight line, and may then run indifferently in the top or bottom benches of the bed.

Gangway Timbering.

Very few gangways are timbered with square timber, round timber being so much stronger and cheaper that it is almost always preferred.

I find that great difficulty is experienced in obtaining timber of sufficient size for good substantial work, and that timber from twelve to fourteen inches is often used where eighteen-inch stuff is really needed.

An average of a large number of measurements of what were commonly considered fair specimens of gangway timber gave somewhat less than thirteen inches, and this may be considered to fairly represent the average size (small end) of the large timber now used.

Much larger timber is, however, obtained and is, indeed, absolutely necessary at many collieries. Timber under fifteen inches is not considered worth much for gangway work in the mines of the Lehigh Coal and Navigation company, where, as elsewhere, constant effort is made to obtain a certain amount of eighteen-inch stuff.

The method of fitting the collar upon the top of the leg is shown by several of the preceding illustrations (Figs. 20 to 26) and on a larger scale by Fig. 27.

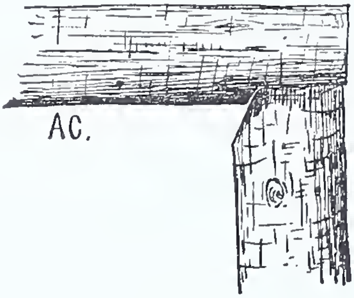


Fig. 27.

The angle at which the collar and leg are joined varies with the style of timbering. For the ordinary upright frame it is usually three or four inches to one foot.

After the legs have been placed in position with the collar resting upon them they are wedged in place and securely fastened by flat wooden wedges as shown in Figs 20 or 21, or by round wedges or lagging as shown by Fig. 28, which also shows a different method of cutting the joint.

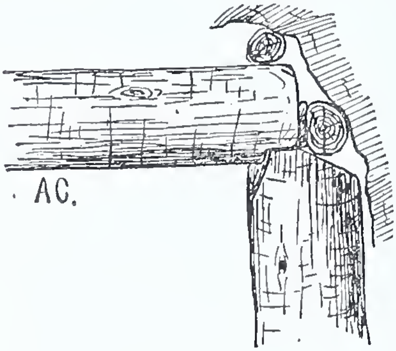


Fig. 28.

The lagging is then driven in behind the legs and over the collars, as shown by Figs. 20 to 26, and if the roof is very poor so that it falls at the face and needs more substantial support at certain points than is afforded by any single set of timbering, long poles from three to five inches in diameter are driven in over the collar, and these are of sufficient length to lap over two sets of timbering. This is termed "fore-poling," from the fact that these poles project in front of the last set of timbers, and is especially useful in reducing the miner's risk from roof-falls at the face of the heading.

In addition to the wedges inserted above the collar, the legs are frequently braced horizontally as shown by Fig. 29.

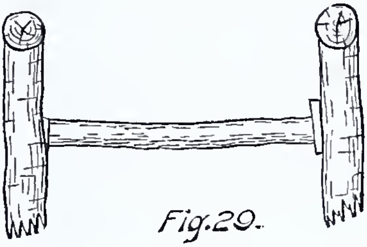


Fig. 29.

In seams nearly vertical and in which the lateral pressure upon the timbering may equal or exceed the vertical pressure, it would seem to be advisable to make the joint between the leg and collar after a design differing from any of those shown in the above illustrations, but as a rule the dip influences the direction of pressure on the timbering only when or after a "squeeze" or "crush" has taken place, and when this occurs no timber can be expected to stand.

Timber used for double track gangways or turnouts without a center-prop is of the largest possible dimensions. When the roof is poor, stuff at least eighteen inches thick should be used, and twenty, twenty-two, and even twenty-four inch timber is sometimes seen in such places.

The lagging commonly used for gangway work is round stuff from three to six inches in diameter; thicker material is used, but not often.

The *life* of mine timber varies between extremely wide limits. In some of the mines of Continental Europe timber is still standing in a good state of preservation apparently unchanged since it was placed in position one or two centuries ago, and in some of our anthracite mines timber thirty years old is still in a fair state of preservation—but this is the exception, not the rule.

As a rule the life of mine timber in the anthracite regions is limited to a few years at most, and at some collieries the average life of all the timber has been estimated at less than one year.

Several causes contribute to hasten the decay of timber in anthracite collieries, prominent among which are warm, damp air, dripping water, running or standing acid water, the presence of air containing gases, and the changes from dry to wet and from a wet to a dry condition of the mines following corresponding changes in the weather.

I do not know that any systematic attempts have been made in this country to prevent the rapid decay of mine timber by preservative or water-proofing material applied to the timber before it is placed underground, but in England, Germany, and France such experiments have been made and with very satisfactory results.

Tunnel Driving.

As already described in the chapter on the methods of opening coal, tunnels have been largely used as a means of developing beds lying above water level, but at present they are chiefly employed in opening workings by driving across the intervening measures from a bed already developed by shaft or slope workings.

A fine example of tunnel development is shown by Fig. 30, which is a section taken through the Nesquehoning tunnel at Lansford, in the Panther Creek basin, here reproduced on a small scale from Cross-Section Sheet No. II of the Panther Creek geological atlas.

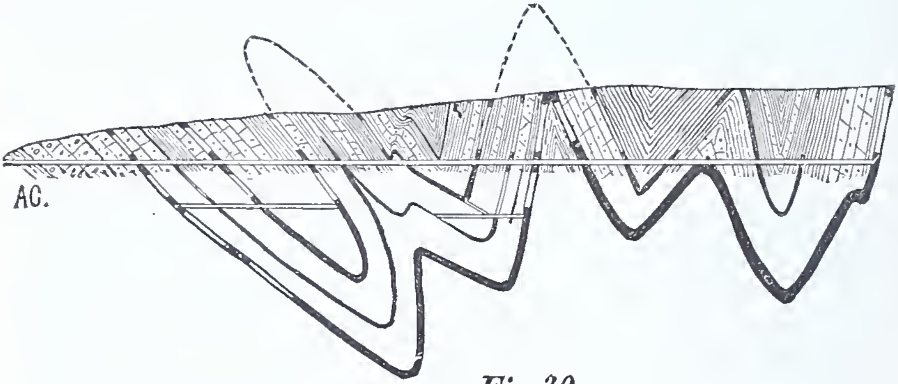
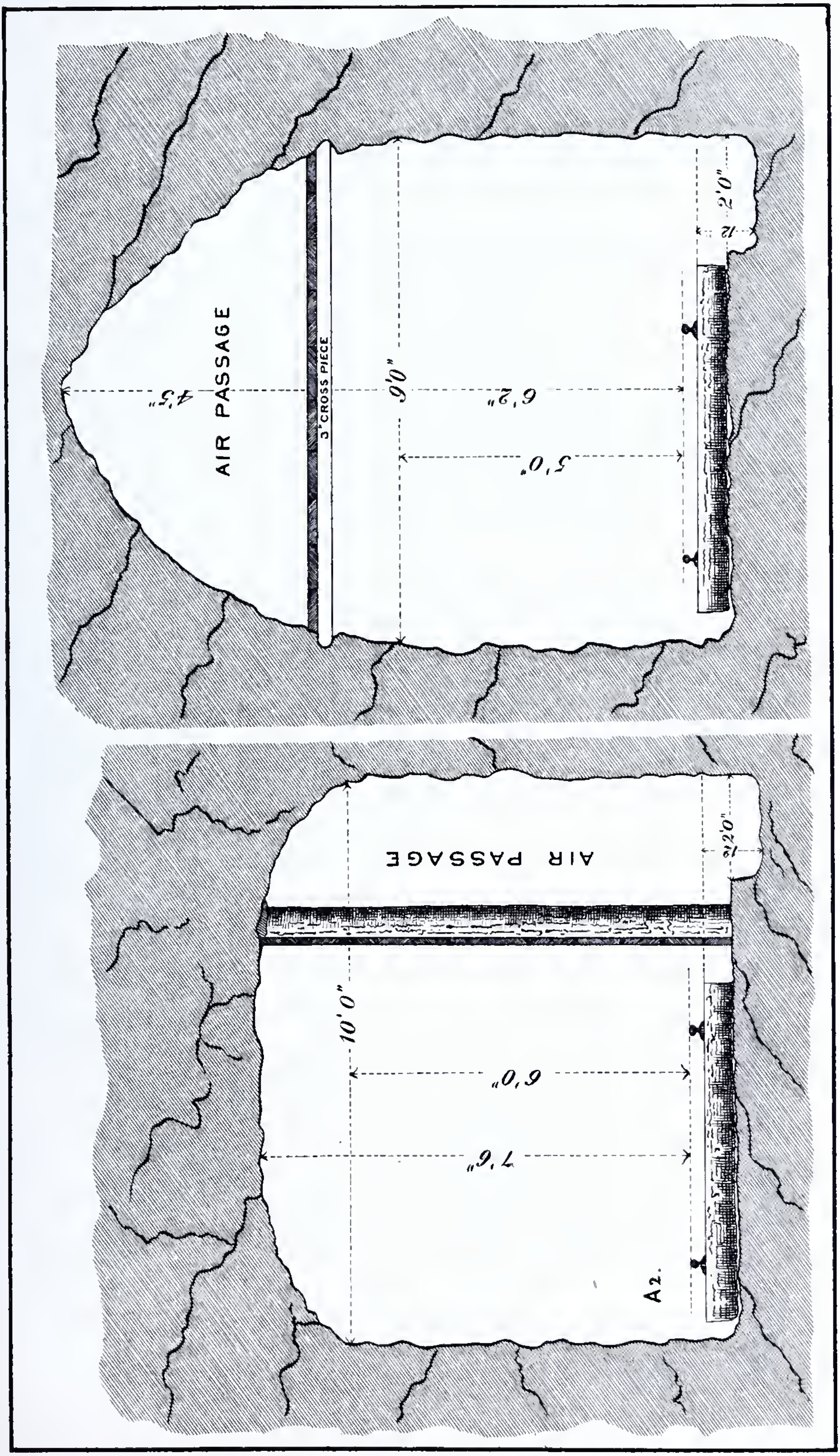


Fig. 30.


AIR PASSAGE
3' CROSS PIECE
4'5"
9'0"
6'2"
5'0"
2'0"
12
AIR PASSAGE
2'0"
10'0"
6'0"
7'6"
A2.

Throughout the Panther Creek basin this method of development by tunnels has been largely used.

Underground tunnels are common throughout the anthracite region, as shown by the maps published to accompany the geological report, but probably in no other district is the geological structure so well suited to this method of development as in the Panther Creek basin.

When new workings are opened on a bed by means of underground tunnelling from a developed bed, as shown by Fig. 51, the workings can only be developed to a limited extent by means of the single opening, the law requiring a second opening to secure the proper ventilation of the working, and to supply another exit for the miners in case of accident.

When the coal developed lies at a shallow depth beneath the surface, the second opening is often obtained by driving an airway (used also as a man-way or traveling-way) up to the outcrop; but when the area thus developed lies at a considerable distance from the outcrop, or when the coal does not come to the surface, a second tunnel is driven at some distance from the first, to supply the necessary exit.

Two cross-sections of tunnels are shown by Page-plate No. 9, reproduced from Report A^2.

In the left hand drawing the airway is carried on one side of the tunnel, and is constructed by a brattice carried on a row of props. The tunnel is about ten feet wide by about eight feet high. In the second form the airway is above the gangway, and the tunnel is driven in the form of a gothic arch nearly twelve feet high and nine feet broad. Tunnels of this shape are seldom used.

As the airway is usually but a temporary device to secure ventilation until a second opening is obtained, the posts and brattice are commonly removed after the new workings are fully opened, and by widening out or driving the tunnel about fourteen feet, the space occupied by the airway can be made to carry another track, thus making at a comparatively small additional cost a two-track opening.

Table Showing Cost of Tunnelling in Lehigh Region in 1878.—From Mine Inspector's Report.

	NAME OF MINES IN WHICH TUNNELS WERE DRIVEN.	DIMENSIONS OF TUNNEL.			INTERVENING STRATA.		Thickness of coal seam in feet.	NUMBER OF ROCK MEN.			TIME OCCUPIED.		Total number of days, of twenty-four hours, in operation.	EXPENSES.		
		Length—in feet.	Height—in feet.	Width—in feet.	Inclination.	Thickness—in feet.		Contractors.	Miners.	Laborers.	When commenced.	When finished.		Labor.	Powder.	Steel.
1	Cranberry, ...	447	7	7	26°	196	6	1	2	6	July 26, 1877	Aug. 18, 1878	324	$3,504 17	$315 00	$32 00
2	Sugar Loaf, ...	86	7	7	25°	36.3	6½	1	2	6	Feb., 1878	March, 1878	38	1,219 16	58 50	1 00
3	South Sugar Loaf, ...	54	7	7	50°	41.3	7½	1	2	6	Dec., 1877	Jan., 1878	28	515 84	36 00	95
4	East Sugar Loaf, ...	131	7	7	33°	71.3	7½	1	2	9	July 1, 1878	Oct. 25, 1878	100	1,395 15	213 06	29 06
5	Coleraine, ...	162	7	9	35°	92.9	4	1	9		Nov. 1, 1877	April 1, 1878	130	2,337 63	194 40	31 60
6	Spring Mountain, ...	150	7	10	68°	139	7½	2	. .	6	April 10, 1878	Sept. 30, 1878	129	2,071 00	357 00	129 00
7	South Sugar Loaf, No. 2. Tunnel, . .	58	7	7	50°	44.4	7½	1	2	6	Dec., 1877	Jan., 1878	35	533 01	45 00	1 89
8	Room Run, (shaft, ...	270	8	11	90°	270	50	1	6	8	April 3, 1876	Dec. 24, 1876	182	5,612 00	432 00	128 00
	Totals, ...	1 358			. .	891.2		9	25	47	. . . .	. .	966	$17,187 96	$1,650 96	$353 50

Table continued.

	Name of Mines in which Tunnels were Driven.	Expenses.			Price per lineal yard paid to contractors for—			Miners wages at the time the tunnel was being driven—per week.	Cost per—			Progress of tunnel per month (of 26 days) in—		Remarks.
		Supplies—oil, wick, &c.	Smithing.	Railroad iron and sills.	Rock.		Coal.		Lineal yard.	Cubic yard.	Total cost of tunnel.	Lineal feet.	Cubic yard.	
1	Cranberry, ...	$152 91	$200 00	$243 09	$33 25	$20 96	$3 92	$9 42	$29 85	$5 38	$4,447 17	35.87	65.10	Sandstone and slate.
2	Sugar Loaf, ...	17 15	32 62	46 78	44 75	27 00	4 00	9 45	47 97	8 81	1,375 21	58.90	106.90	Sandstone and slate.
3	South Sugar Loaf, .	12 63	26 96	29 36	34 75	30 00	5 50	8 93	34 54	6 34	621 74	50	90.74	Sandstone and slate.
4	East Sugar Loaf, . .	57 80	50 85	74 24	39 50	. . .	. . .	9 67	41 62	5 94	1,817 16	34.03	79.39	Rock medium hard, sandstone.
5	Coleralne, ...	80 38	300 00	65 00	45 00		. .	9 67	55 72	7 96	3,009 01	32.40	75.60	Rock extremely hard, sandstone.
6	Spring Mountain, . .	70 00	250 00	65 00	50 00	. . .	. . .	9 45	58 84	7 56	2,942 00	30.24	78.40	Rock extremely hard, sandstone.
7	South Sugar Loaf, No. 2 tunnel, ...	15 79	22 51	31 42	33 50	22 00	5 50	8 93	33 60	6 17	649 62	42.96	77.97	Sand rock and slate.
8	Room Run, (shaft,) .	128 00	389 00	131 00	70 00	.	.	12 36	75 78	7 75	6,820 00	38.57	125.72	Rock extremely hard, sandstone.
	Totals, ...	$534 66	$1,271 94	$682 89	. . . .	. . . .	. . .	. . . .	. . . .	. . .	$21,681 91			

The gutter is commonly located on the floor of the airway.

Tunnels to develop very gaseous beds of coal, are commonly driven from twelve to sixteen feet wide. Double track tunnels are preferred under such circumstances and the size inside the timbering is rarely less than seven feet by twelve.

Tunnels of much smaller size are in use at collieries where the workings are small, and but little trouble is experienced from gas, or where the ventilation is effected through other channels. Under such circumstance tunnels only six or seven feet wide and seven feet high have not been uncommon in the past, especially when the rock to be pierced was very hard; but openings of this size are rarely driven at present.

Cost of Tunnel driving.

The cost of driving depends somewhat upon the cross-section of the tunnel. For tunnels of average cross-section the cost in hard rock may be considered to average from five to eight dollars per cubic yard, in moderately hard rock from three to six dollars per cubic yard, in shale or slate from two to three dollars per cubic yard, and in very soft slate from one dollar and a half to two dollars and a half per cubic yard. The average cost of driving through alternately hard and soft rock varies from three to five dollars per cubic yard.

There can be no doubt of the economy of steam or compressed air drills for work of this class, and the only reason that they have not been generally adopted is the fact that when the work is finished there is seldom any use for the air compressing machinery.

This latter can, however, be used with good results in operating underground slopes and direct-acting steam pumps, as at the Mahanoy city colliery where a double engine, supplied with air by two double compressors of the Rand type, is used to operate an inside plane extending from the second to the third lift; and at Kohinoor colliery, where four engines and a fan, all underground, are run by air supplied by a double compressor of the Rand type, two Burleigh compressors are held in readiness to supply the place

Rand compressors in case of accident or during re-
to the latter.

s subject will be considered at greater length in the
er on underground slopes.

hough for the above reasons not yet generally adopted,
drills for tunnel driving have obtained a firm foothold
anthracite regions, and with the growing use of com-
d air for operating underground machinery they will
less occupy a more prominent place in the near future.

nost all forms of power drills manufactured in this
ry have been used in the anthracite, but the Ingersoll,
and Burleigh seem to be preferred to other makes.

an eight by ten or eight by twelve tunnel two drills
e worked, and from forty-five to sixty feet a month
e driven in rock of average hardness. In soft slate or
the power drills probably do not give any better re-
than can be obtained by ordinary hand work, and it is
iving through very hard rock that these drills appear
e best advantage.

r hard rock, "Rend-rock" and other forms of the
er explosives are used, and fired by a detonating fuse
y electricity. In hard rock at least four holes on the
om and three to blow down the top are necessary, but
n eight by twelve tunnel nine or ten holes are some-
s drilled before the squaring up is finished. In soft
four or five holes are often all that are needed.

hen the rocks are dipping towards the mouth of the
nel, the top holes are best fired first and the bottom
vn up afterwards, when dipping away from the mouth
bottom holes are fired first. When the rocks are nearly
ical a central group of converging holes is first fired,
the sides, top and bottom crushed in by a series of per-
eral holes. The cleavage may exercise a greater influence
n the dip in some cases, and the position of the holes is
n determined independently of the direction of dip.

unnel driving is usually done by contract, and the plan
driving varies with the views of the individual contrac-
s. Some always keep the top in advance, some the bot-

tom, others alternate, the top and bottom being in adva alternately, while in some cases one side is kept in adva

Disregarding any assistance offered by the bedding cleavage of the rock, the better plan is probably to k the centre near the bottom ahead, but the cleavage and b ding are such important factors in work of this class th is not possible to determine in advance the plan that give the best results in any particular case.

Tunnel Timbering: Comparatively few tunnels in anthracite regions require timbering throughout, and very large number contain no timbering whatever.

Tunnels driven in from the surface are usually timbe for a short distance from the mouth, or until hard rock reached; but underground tunnels are timbered only points where soft coal or a loose friable rock is encounter

Tunnels of the former class occasionally pass throu loose earth carrying a considerable amount of water, as the L. C. & N. Co. No. 9 Colliery, and a species of quic sand has been encountered at some openings of this class

Under such circumstances a completely framed timberi is needed, consisting of a bottom sill, two legs, and a coll usually made of square timber similar to the slope timbe ing shown by Fig. 12 and Page-plate No. 10. The timbe are set not more than two and a half or three feet apa and sheathed with a heavy plank, often reinforced by for poling.

The timbering ordinarily used for tunnel work is pr cisely similar to the gangway timbering shown by Figs. 2 21, and 22, and needs no further description.

Ventilation:

In short tunnels, and occasionally in tunnels of consid erable length, ventilation is effected by a small blowing fa and board pipes; but the plan giving the best results that (already described) shown by Page-plate No. 9, i which the tunnel is divided by a brattice into an intak and return airway.

CHAPTER VII.

The Mining Plant at the Surface.

In no feature do anthracite collieries differ from each other as much as in the arrangement of the plant above-ground. This is due in part to generic differences in the structures that go to make up the plant, and in part to the topography of the surface on which these structures are erected, but principally because the mine superintendents and engineers throughout the region, apparently have not sufficiently appreciated the fundamental principles by which the best arrangement of all such plants should be determined. At some of the more recently opened collieries,—notably those of the Lehigh Valley Coal company,—the arrangement of the plant is all that can be desired ; but at nearly all of the older collieries, and not a few of the newer openings, the various structures composing the plant seem to have been located at random, and with no view to the harmonious working of the whole as a unit.

However, at the older collieries this has doubtless been brought about by the remodelling and enlargement of old plants, and the addition of new structures, to meet the requirements of an increased output.

At an anthracite colliery the plant at the surface usually consists of the structures enumerated below, and the object to be attained in determining its arrangement, is the harmonious working of the whole as a unit without destroying the individuality of any of its members. Thus we ordinarily have :

1. The Head-frame, or, if weather-boarded, the Head-house.
2. The Winding Engine, Drums, etc.
3. Boilers and coal-bins.
4. Breaker Engine.
5. Breaker.

6. Culm or dirt, or waste and rock banks.
7. Dirt plane?
8. Pumping Engine.
9. Fan.
10. Breaker Engine.
11. Breaker Hoisting Engine?
12. Railroad tracks.

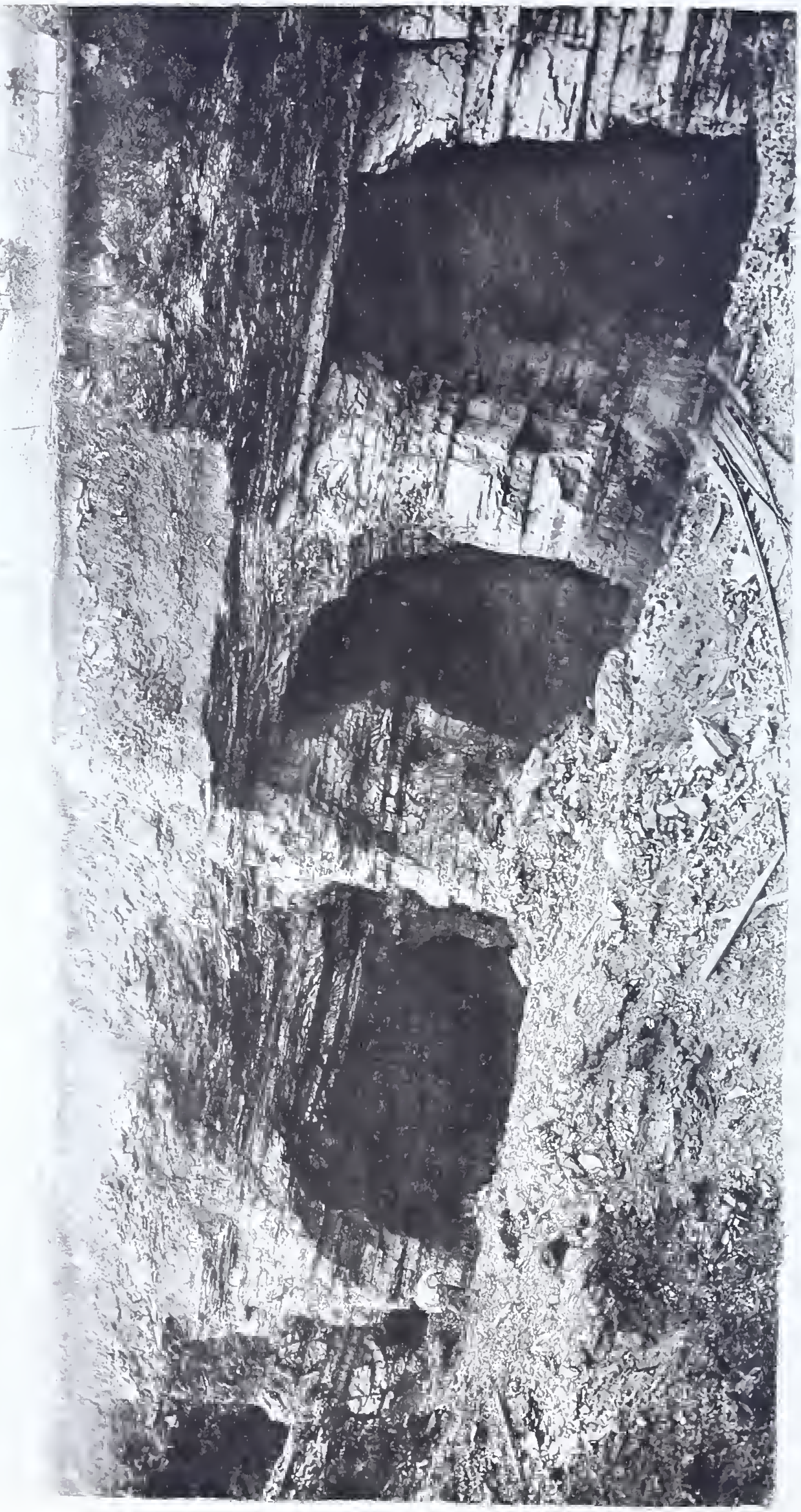
It is necessary to refer first to the old plan of building the breaker over the shaft. When this plan is adopted, the breaker, head-frame, and engine-house (?) are merged into one, and the cars are raised directly to the top of the breaker. The first cost of such a plant may be less by from five to ten thousand dollars than a plant in which these are erected as separate structures, and a saving in the cost of operating such a plant is also effected.

On the debit side of the account, however, we have created a risk that may at any time wipe the whole structure out of existence, and at the same time greatly injure or ruin the mine.

When this plan is adopted the engine-room is usually, if not inside of the breaker, a contiguous structure, and as in this room, cotton-waste and other oily and extremely inflammable material may always be found, the risk of fire is by no means small. When the engine-room is in the breaker, the risk is greatly increased by its location, and by the probability of cotton-waste being thrown or blown by the draught into out-of-the-way places where spontaneous (?) combustion may occur. In addition to these probable causes of fire, there are within the breaker itself many possibilities of conflagration, the principal of which are hot journals, oil, and cotton-waste, and in addition, the risk from incendiarism.

The damage done to the shaft and the probability of a mine fire resulting from the burning of the breaker, in addition to the risk of great loss of life in the mine under such circumstances, are sufficient reasons why such a plan should not be adopted, but there are others equally if not more forcible, and these may be briefly summarized.

In gaseous mines such an arrangement is especially dan-



WORKINGS AT HOLLYWOOD COLLIERY NO. 1,-- LOOKING SOUTH. (PICTURE REVERSED.)

ARTOTYPE, E. BIERSTADT, N. Y.

gerous, as in the event of a serious explosion in the mine, the breaker may be greatly injured if not destroyed, and many of the breaker hands (numbering from one to two hundred) killed and wounded; again, in case of a mine fire involving the shaft, or of fire in the shaft alone, the destruction of the breaker is nearly certain.

In addition to the risk from fire and explosions, the damage that may result to the breaker and the risk to life from any accident (whether from breakage or over-winding) to the winding machinery is not to be under-estimated.

Notwithstanding all these risks incurred by erecting the breaker over the shaft, many private operators and incorporated companies, adopt this arrangement and accept the accompanying risk, rather than incur the additional expense entailed by constructing and operating the breaker, head-frame and engine-house as separate and independent structures.

A compromise between these two plans has been adopted by the D. & H. Coal Co., as shown by Page Plate 43 in which the head-frame is built some little distance from the breaker, and connected with it by a trestle-work. The cars are raised at once to the level of the breaker-dump and run over the trestle to the tip. This design is free from most of the objections that pertain to that in which the breaker is placed directly over the shaft, but *the risk is greatly increased by housing in the head-frame and trestle.*

In case of a mine fire involving the shaft this head-house and the covered trestle would act as a flue and draw the fire over into the breaker, and in case of fire in the breaker the head-house would probably be involved through its connection with the covered trestle. To reduce the risk to a minimum the head-frame and trestle should be built of iron, or of large heavy timbers, (roofed in if considered absolutely necessary) without weather boarding of any kind,—in other words an open skeleton structure that will not readily burn.

At collieries at which the coal is raised through slopes, these objections do not apply, as the breaker is then placed in front of the slope-mouth; the main structure standing fifty feet or more from the opening, and connected with it by an inclined plane (forming a continuation of the slope) built as an open trestle.

Some of the same objections that apply to the location of the winding engine within the breaker, apply also to the prevailing practice of erecting the breaker engine in the lower part of the breaker; but the large number of breakers destroyed by fire directly or indirectly traceable to the breaker engine-room, is effecting a rapid change in this particular. A considerable number of breakers are now run by belting or wire-rope transmission of power from an engine located some distance from the main structure; as shown by the plan of the Exeter Colliery plant, Atlas Sheet No. IV.

Yet this method of operating a breaker is open to objection on account of the almost complete separation of the engineer from the breaker hands. This objection could be removed by a system of signal wires leading from different portions of the breaker to an alarm-gong in the engine room, which would enable the breaker hands, in case of accident, to notify the engineer promptly to stop the machinery.

Head-houses are objectionable from their liability to destruction by fire, and for this reason, if for no other, unhoused head-frames are preferred when the breaker is not built over the shaft mouth. Iron frames are of course more durable than timber structures, and are not quickly injured by a shaft fire; but on account of their relatively greater cost they have been erected at but few collieries.

It is certainly poor engineering practice to inclose an expensive winding engine in a frame structure. Engine rooms at large collieries newly opened are now usually built of brick or stone, and separated by at least ten feet from the boiler house.

The boiler houses commonly built are frame structures—often the rudest kind of a shed, built of slabs and waste boards; but quite respectable stone and brick housings are seen at some collieries. The character of the boiler house, aside from its inflammable nature, is a matter of small importance.

When the plant contains a winding engine, a breaker winding engine, and a breaker engine, besides feed-water pumps and a mine pumping engine, it is evident that all of them can be most economically erected and operated in one structure; but as the presence of other machinery may divert the attention of the engineer in charge of the main winding engine from his duties, and the presence of other engines, and the increased number of signals from gongs, etc., may be the cause of more or less grave accidents, the winding engine is usually isolated from other machinery, either by placing it in a separate structure, or in a room not communicating with the room containing the other machinery.

The breaker engine, breaker hoisting engine, mine pumping engine, and feed-water pumps, may all be housed in one structure, except when the mine pump is of the direct-acting Bull pattern, as at the Exeter colliery, (see Atlas Sheet No. IV.)

When the topography of the ground permits the location of the breaker at a point low enough to bring the top of the breaker (the cradle dump) on a level with the mouth of the shaft, a considerable sum can be saved annually; but such an arrangement can rarely be effected, and when this is done, the breaker is commonly at a considerable distance—in one instance a mile and a half—from the coal opening.

In all cases the breaker is so located that a railroad track may be constructed, with ample grade sidings, and of sufficient fall to handle the cars by gravity.

While there are many reasons (already enumerated) why the breaker should be located at some distance from the shaft opening, the handling of the mine cars, and the impossibility of moving them both ways by gravity, make it desirable to reduce this to the least possible distance consistent with safety from fire, etc.

The loaded cars are commonly transported to the breaker on a track having sufficient fall to move the cars by gravity, and the empty cars are drawn back to the shaft by mules; but the grade is sometimes so slight that mules are required on both the loaded and empty tracks.

To reduce to a minimum the loss of heat and steam by radiation from the steam pipes, it is of course advisable to have the boilers located as near the engines as possible.

Where coal is as cheap as at an anthracite colliery, especially when buckwheat coal is burnt under the boilers, economy of fuel is not an important factor, but the water used throughout the anthracite regions is often of a miserable quality for boiler use, and it is especially desirable on this account to economize in the use of steam, and use as few boilers and as little water as possible.

Hence the economy of the arrangement shown by Atlas plate No. IV in which the boilers are placed between the two engine-houses, and the steam connections are correspondingly short.

By locating the base of the boilers a few feet below the average level of the surface, the coal-bins are filled by a car dumping the coal in bins directly in front of the fire boxes; the same object is accomplished by building the boiler-coal track on a tressel (near the boiler house) so that the coal may be dumped into shutes and run directly to the firing floor, in front of the boilers. This car brings the coal down from one of the buckwheat coal pockets in the breaker to the boiler coal-bins.

In the location of the winding engine with reference to the head-frame, important problems seldom occur. It is found necessary, first, to place the winding engine at a certain distance to give sufficient lead so that the rope will coil regularly on the drum with a minimum amount of friction; and second, not to increase the lead to such an extent that the rope will be subject to violent oscillations. This latter difficulty is sometimes overcome by placing bearing pulleys between the drum and the sheaves. Such pulleys run free on a shaft of sufficient length to allow them to travel laterally with the rope as it coils or uncoils from the drum.

When the fan is located directly over the airway it is subject to more danger from fire than when located a few feet away from the shaft mouth, but when fans are not placed directly over the up-cast they are generally placed so very close to it that the risk of damage by fire is almost as great.

The principal reason, however, for locating the fan on one side of the upcast is to permit the use of this latter for the pumps and pump rods, and also in some cases, as at the Pottsville big shafts, the Exeter second opening, etc., to allow this opening to be used as an occasional hoisting compartment.

Until within the last few years the culm or coal dirt, the bony coal picked from the coal in the breaker, and the slate and rock coming from the mine were all deposited together in one heap or series of heaps, but the possible utilization of the culm or fine coal, either by burning in fire-boxes constructed for this purpose, or by manufacture into artificial fuel has given to the culm as it lies in the banks a certain prospective value, and to enhance this value and reduce the future cost of utilizing this refuse, two or three distinct kinds of refuse heaps are now common.

The fine coal known as culm, coal dirt, dirt, etc., that passes *through* the buckwheat screen mesh (or when no buckwheat is made, through the pea coal screen mesh) together with all the fine stuff made in the pockets and larger screens and collected at the lip and telegraph screens, is collected in separate pockets in the breaker and finally deposited in a heap known as the culm-bank or dirt-dump.

The rock and slate coming from the mine is carried directly to a heap known as the rock-dump, and the bony and slate picked from the coal in the breaker after being collected in pockets in the breaker is dumped on the slate or bony coal-bank; but the rock and bony coal are not unfrequently stored in a common heap, and at some collieries the bony coal is all recrushed in pony rolls and passed into the smaller sizes of coal, in which case the rock-dump contains nothing but slate and rock, with some coal adhering to the lumps.

In determining the location of the rock and culm-dumps the topography of the surrounding area exercises a governing influence, but it is always advisable to avoid depositing this refuse over the outcrop of a workable seam. When these heaps catch fire there is more or less danger of the fire extending to the outcrop coal and into the mine.

These heaps increase rapidly in size and often become of almost unmanageable dimensions, occupying ground needed for other purposes. The land thus occupied is, however, seldom of any value for agricultural purposes, except in some parts of the Wyoming region.

At slope collieries opened on the edge of the basin, the ground generally falls away rapidly enough to give ample dumping space within a short distance from the opening, but in many cases this descent is utilized by building the breaker below the mouth of the opening, and the culm is taken out at the foot of the hill.

Under these circumstances and generally at shaft collieries, or at slope openings on comparatively low ground, dumping height is obtained by an inclined plane, usually known as a dirt-plane.

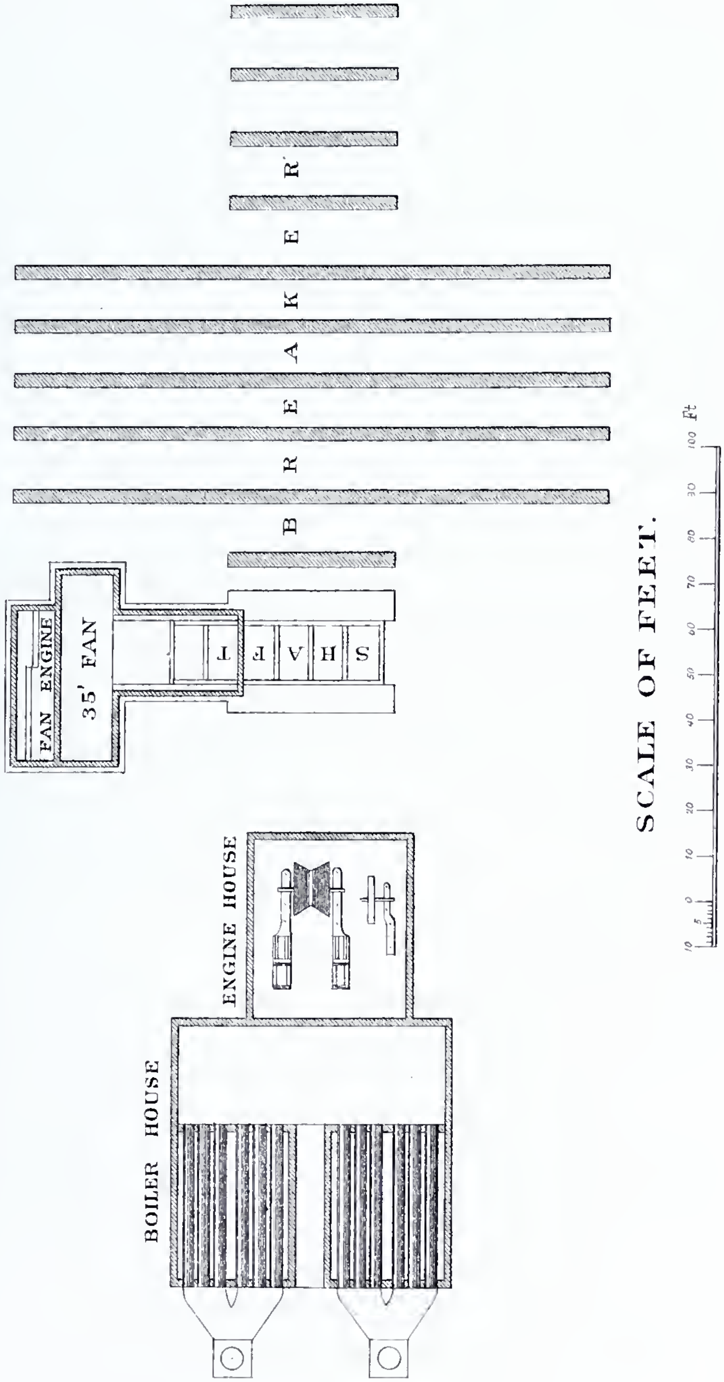
Page plates 49 and 50 give some conception of the size these heaps attain and the amount of ground some of them cover. I have estimated that there is enough of this material to cover ten square miles of ground forty feet deep, or almost enough to form a pyramid one mile square at the base and twelve hundred feet high.

When no dumping ground at a lower elevation than the breaker is available and a dirt-plane is necessary, this should be located as near the breaker as circumstances permit; but that it should not face the breaker or any other important structure that may be injured by cars breaking loose on the plane, is equally apparent.

When the coal is washed and a large quantity of culm is carried off by the water through a sluice or flume, the nearest creek or water-course is often made the receptacle for this material, but as such channels seldom carry a sufficient quantity of water to sweep away this sediment the streams are quickly clogged up by it, and after a time strew it in a thick deposit over the lower portions of the valleys, giving rise to suits for damages, etc.; the construction of large settling pools is therefore often necessary.

A good example of a compact mining plant is shown by Page plate No. 11, which represents the location of the

PLAN OF THE HOLLENBACK PLANT.



breaker, engine, boilers, etc., at the Hollenback shaft colliery.

In the preceding pages the objections that may be urged against such an arrangement of the plant and its actual disadvantages have been considered in detail, but its advantages have not been stated; in other words, its faults have been criticised but its merits ignored. This may seem unjust, but the advantages of a compact plant are so obvious that I have unwittingly neglected them, much in the same way that a mathematician often omits quoting an axiom in proof of a proposition when the application is obvious. It may, however, be well to enumerate the principal advantages accruing from compactly constructed plants of this type.

1. Less first cost of the plant.
2. Less ground occupied by the plant.
3. Mine cars handled but once.
4. Economy of steam power.
5. Proximity of the breaker engine to the breaker machinery.
6. Less cost of operating the whole plant.

At the Exeter colliery a plant of totally different type is seen, as illustrated by Atlas sheet No. IV. The breaker is located about three hundred feet from the shaft; the winding engine is housed by itself, and a separate engine-room is provided for the breaker hoisting and breaker driving engines, the power from the latter being transmitted by a rope running on sheaves thirteen feet and six inches in diameter; the boiler-house is independent of all other structures, and the head-frame is an open piece of timber framing.

This plant is admirably adapted to accomplish the objects enumerated in the first few pages of this chapter, viz: The reduction to a minimum of the risks to the plant and to the mine, of destruction by fire or by mine explosions, and of damage from breakage or over-winding. At the same time, the construction of plants embodying the principles on which this has been designed, while preserving the individuality of each structure, is not incompatible with the harmonious working of the whole as a unit.

Any discussion of this subject inevitably culminates in the problem already indicated by the considerations advanced above, and this is resolved into a question of economy and risk to life and property which must be solved by each individual operator or mine superintendent according to his own judgment.

A compact, cheaply built and cheaply operated plant, with certain risks to life and property; *versus*, a plant consisting of disconnected structures, more costly, but with less risk to life and property.

Chapter VIII.

Mining Systems.

Coal mining operations are naturally sub-divided into two classes : 1. Surface Mining and 2, Underground Mining.

Surface mining includes both stripping and open-cut quarry work, examples of which are furnished by the old Summit Hill mines, near Mauch Chunk, the Baltimore open cut work near Wilkes-Barre, and the present workings at Hollywood (Page plates Nos. 12 and 13) colliery and Hazleton No. 6 colliery near Hazleton.

As coal beds are readily disintegrated by atmospheric action, the occurrence of localities at which the coal is covered by a moderate amount of overlying rock and soil, and yet is of good marketable quality, are rare. In the bituminous region extensive strippings are almost unknown, but in the anthracite regions the character of the coal and the thickness of the coal beds have, at a few localities, produced conditions extremely favorable to open surface working.

The coal now being stripped at the Hollywood colliery four miles north-west from Hazleton, lies in two canoe shaped basins. The southern basin was worked by a slope, (a gangway being driven along the center of the basin,) before the stripping operations were commenced.

The basin is about one hundred and twenty feet deep with dips of 30° to 40° and the sandstone and shale lying within it is largely fissured and broken so that most of it is easily broken up into fragments of manageable size.

The operation of removing this covering or filling from the basin is accomplished by a shute leading up from one of the breasts into the funnel-shaped excavation shown by Page plate 12. An inclined plane extends from the surface down through this breast to the shute from which an ordinary dump-car is loaded with the débris.

The rock, shale, and slate, all in a more or less decomposed or fractured condition, are loosened up with a pick by laborers located at different points around the inside of the excavation, and all this débris slides without assistance into the loading shute below. As the size of the excavation increases and the angle diminishes the débris is run on sheet-iron. (See the illustration.)

When all the débris has been removed from one portion of the basin the inclined plane is shifted to another breast.

When the overlying material is loose, and can be handled by gravity, as at these Hollywood workings, it will probably pay to remove an average of two or three cubic yards of débris for every ton of coal. The cost of handling material in this way, at one dollar and a quarter per day for laborers, and from two to two and a half dollars for mechanics, will range from fifteen to twenty-five cents per cubic yard.

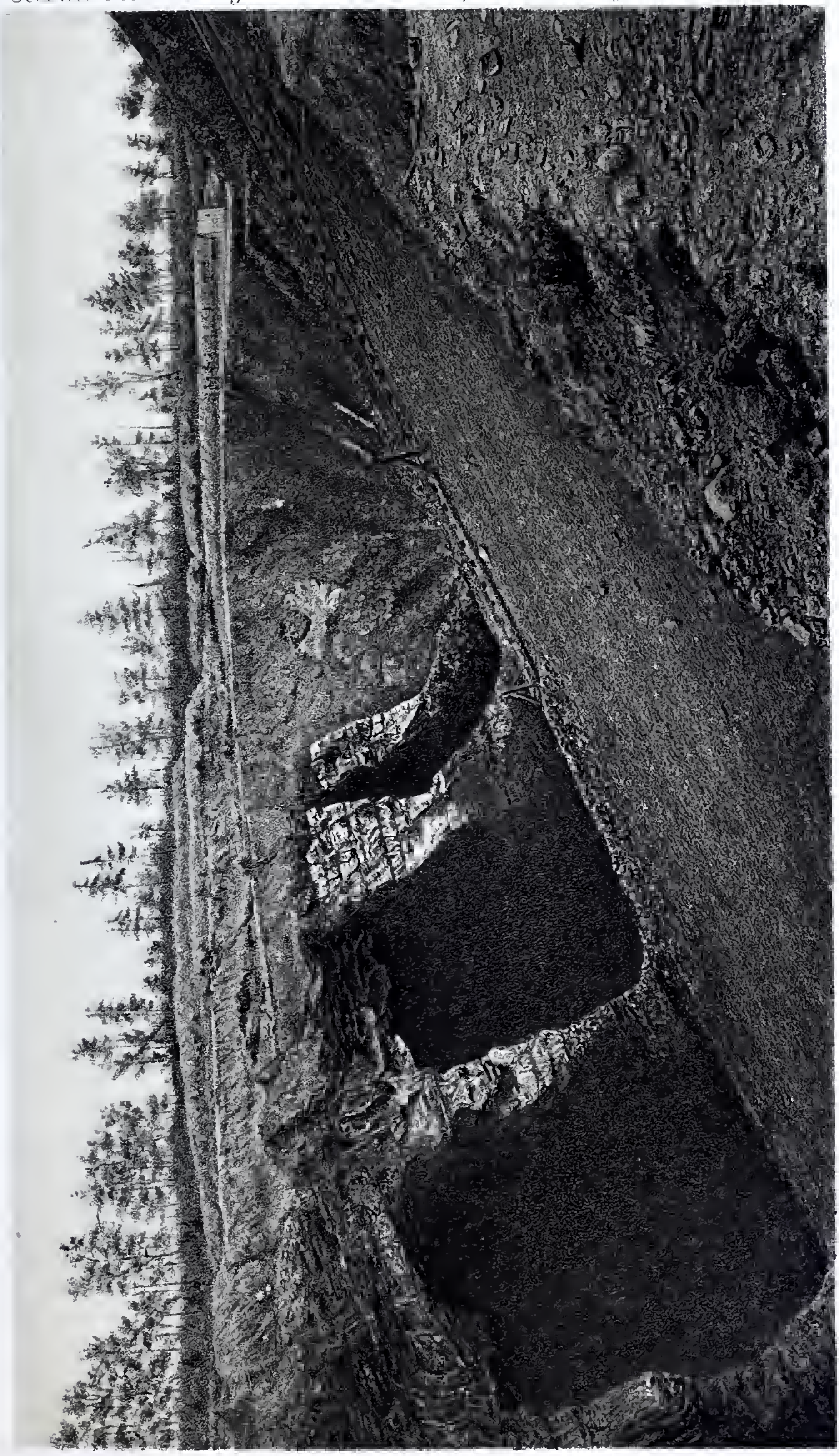
After the stripping is completed the coal will be removed through the gangway driven in the bottom of the trough.

At the Hazleton No. 6 colliery a large amount of flat coal has been exposed ready for mining. The bed is quite thick, lies flat, and carried but from four to twelve feet of cover; yet the upper benches of the bed will furnish much good hard coal.

Operations of this class carried on in the full glare of the sun, are rare, and may well be classed among the curiosities of coal mining; they may, however, become much more common if the land owners of coal lying in shallow basins will consent to reduce the royalty on coal mined by this method. As only about fifty per cent. of the coal is obtained by the ordinary method, and all of it may be recovered by stripping, a royalty of twenty-five or thirty cents per ton will pay the land owner better than fifty cents per ton on the ordinary plan.

The depth of surface that can be profitably stripped has not yet been determined, but the above estimate of two to three cubic yards (yards in depth) to every cubic yard of coal (yards thick) will probably hold good in loose ground.

Coal has also been stripped at Yorktown, Beaver Brook,



STRIPPING AT HOLLYWOOD COLLIERY.

LOOKING WEST.

The breasts are driven of various widths, usually from five to twelve yards, varying with the character of the roof; if possible, of such a width that the roof will be self-sustaining.*

The rooms or breasts are separated by pillars of solid coal (broken only by small cross headings driven for ventilation) from five to ten or twelve yards wide. The pillar is really a solid *wall* of coal separating the working places.

When the object is to obtain all the coal that can be recovered as quickly as possible, the pillars are left thin; but where this plan is likely to induce a crush or squeeze that may seriously injure the mine, larger pillars are left and after the mine has been worked out, the pillars are "robbed" by mining from them until the roof comes down and prevents further working.

In the steeply inclined seams of the anthracite regions the pillar and breast system is employed by working the bed in "lifts," as shown by Page plate No. 14, reproduced here from report A². This will be described in detail in the latter portion of this chapter.

2. *Long-wall system.*

In this system no attempt is made to support the roof of the working places by pillars of coal. It is not applicable to steep dips, nor to thick seams, and has not yet been used in anthracite mining. It is especially adapted to bituminous coals of moderate thickness, dipping less than thirty degrees.

Two kinds of long-wall working are commonly recognized. In the first method the gangways are driven to the outer boundary of the proposed mine and the coal is worked back towards the shaft; the operation is reversed in the second method. These two methods are known by the terms "Long-wall retreating" or "Long-wall withdrawing," and "Long-wall advancing."

In long-wall advancing the roads or gangways are kept

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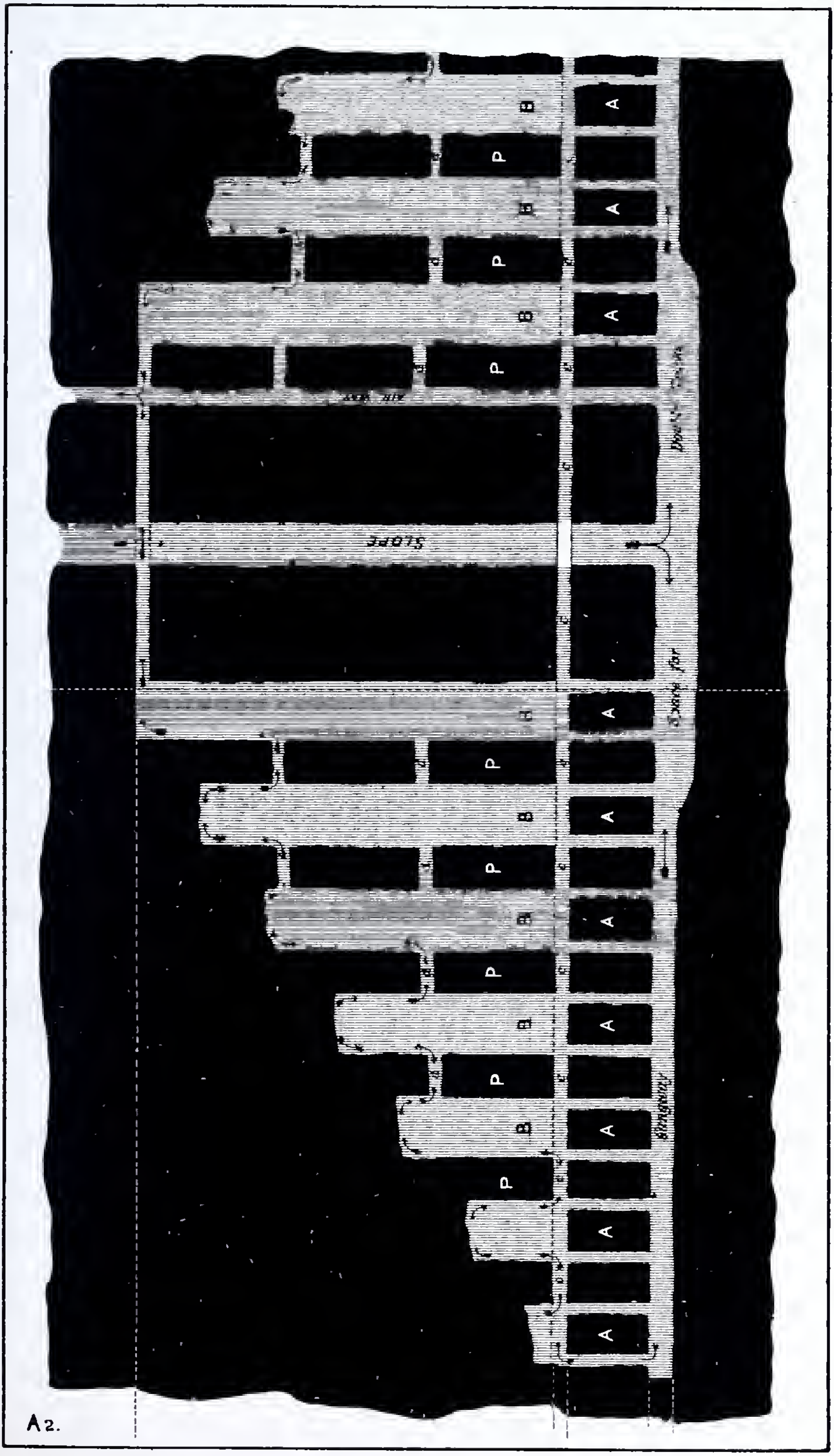
*These are general descriptions, not applying to this system in the anthracite, but also in bituminous mining. The details of this system as used in the anthracite region will be found further on.


SLOPE
AIR WAY
Sump for Double Gangway
Gangway
A
B
P
C
A2.

open and the roof is supported by pack-walls built of débris ("*gob*"). (See page plate No. 15.)

The coal is worked in a long face—hence the name; long-wall—and the roof is allowed to settle down and fill the cavity left by the removal of the coal. When the roof exhibits a tendency to break off close to the working face, it is temporarily supported by timber cribs or "chocks," or by a double or triple line of props. Page plate No. 15* shows these two plans as adapted to long-wall workings in which the roof is of this character. It also shows the pack-walls necessary to keep the gangways open in working by the advancing method. In long-wall withdrawing the main gangways are in solid coal and pack-walls are consequently unnecessary.

The long-wall system of working is now practically in use at several bituminous mines in Pennsylvania, Ohio, Indiana, Illinois, and West Virginia.

I do not know that any attempt has yet been made to use it in anthracite mining. The continual use of powder in blowing down the coal might often bring down the roof and close the working face, but this is by no means certain, and at some localities in the anthracite regions—especially in the Wyoming basin which contains several seams of moderate thickness lying on very flat dips—the conditions are favorable to a trial of this system.

A section across the face of a long-wall working, reproduced here from Mr. Warrington Smyth's work on coal mining, is shown by Page plate No. 16. Two rows of props are shown, the last row being removed as the face advances and set up in front of the first row. In some long-wall workings no props are necessary, the roof arching down to the floor in a long wave-like bend.

Attempts have been made (in England) to substitute iron for wood in propping and timbering. One form of cast-iron prop successfully used in long-wall working is shown by

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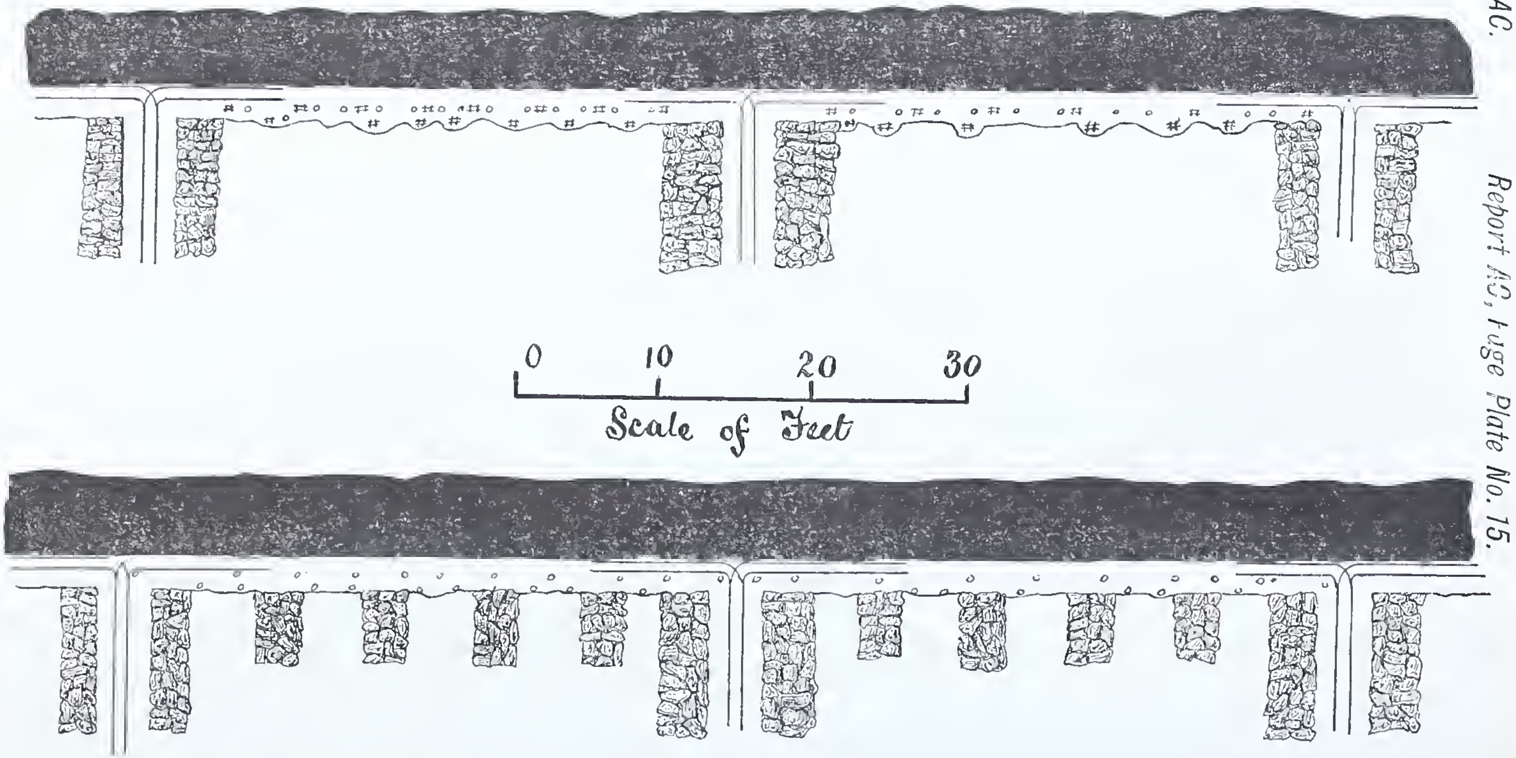
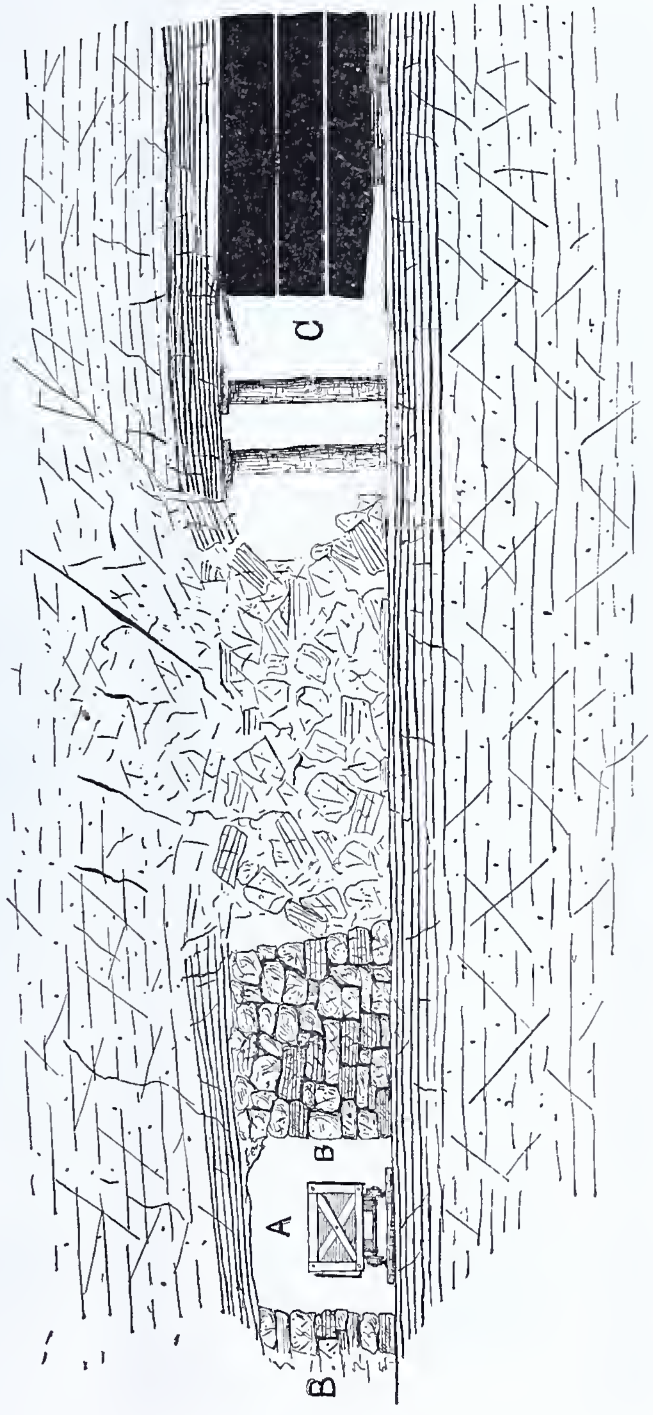
* From the transactions of the North of England Institute of Mining Engineers, Vol. XXV.


Scale of Feet



Section across the face of a Long-wall Working.

Page plate No. 17,* which represents the prop used at the Annesley Colliery in Nottinghamshire, Eng.

These props were used as auxiliary to the pack-walls, which were kept within six or seven feet of the face and about nine feet apart, the roof being very poor. As the roof settled down and obtained a firm bearing upon the pack-walls, the last row of props was removed and placed in advance of the first row.

The props were three inches in diameter inside, and about four inches outside, the metal being somewhat more than half an inch thick ; they were made five feet long and weighed about one hundred and forty pounds.

The illustration shows how the props sink into the roof as the latter settles down upon the pack-walls, but this could not occur in a very hard roof. At the Annesley colliery, after having these props in use four years, the statement was made that not a single prop had been broken, and that they cost much less than wooden propping.

It seems probable that the time is not far distant when wood may advantageously be replaced by iron in the timbering of gangways at mines in which the timber rots so quickly that frequent renewals are necessary. In railroad engineering wood has almost entirely given place to iron in bridges and in trestle works of large dimensions, and the adoption of iron sills (sleepers) is one of the possibilities of the near future. The scarcity and cost of timber suitable for work of this class, and the improvements in blast-furnace practice, whereby the cost of producing cast and wrought iron and steel has been wonderfully cheapened, have effected this change, and it is not improbable that the same causes may effect a like change in the timbering of mines.

A frame composed of cast-iron legs with a wrought-iron cap-piece ought to give good results. If a short cap-piece (collar) is required cast-iron would probably give the requisite strength, but for such purpose cast-iron is open to objection from the fact that it might often snap off without

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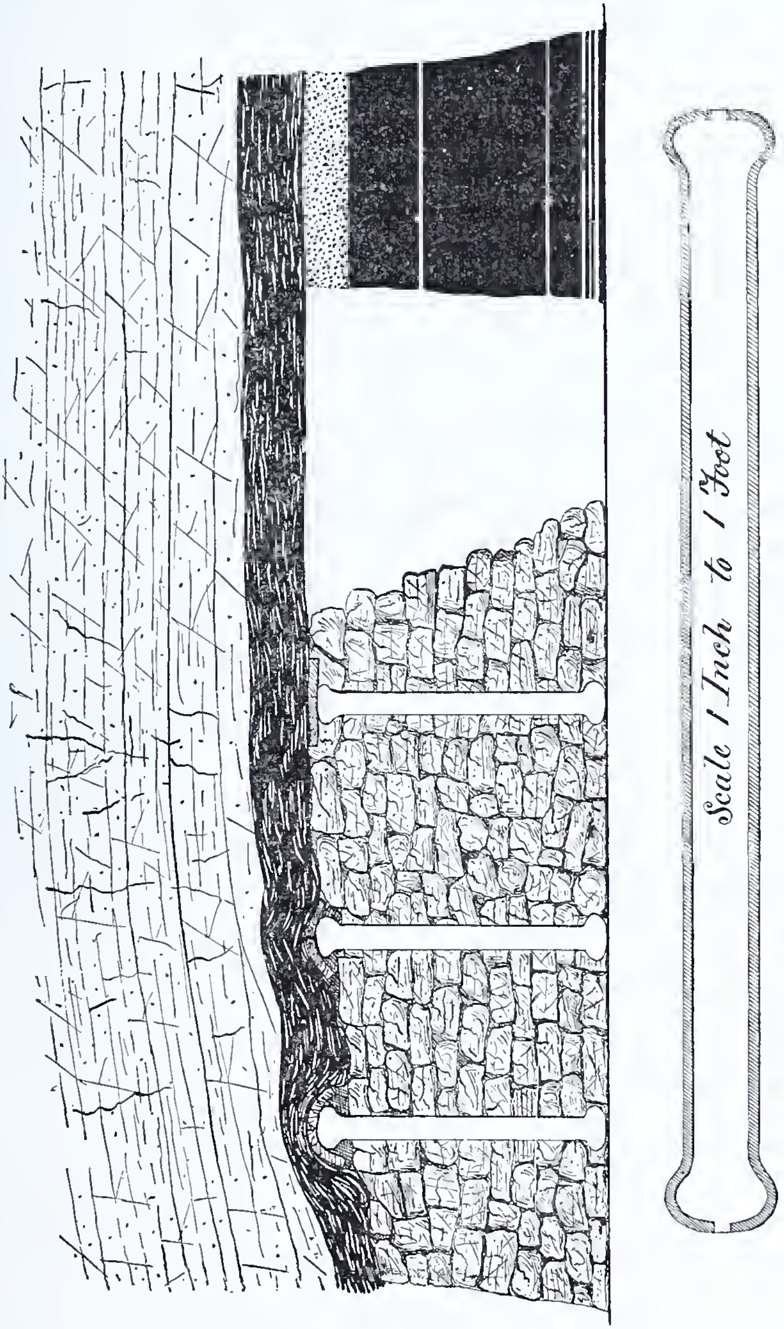
*From the Transactions of the North of England Institute of Mining Engineers, Vol. XXI, (1871.)

Report AC, Page Plate No. 17.



warning, when subjected to increased strain by any settling or loosening of the roof, while wrought iron would adjust itself to the new condition by bending.

3. *The Panel System.*

This system is a modification of the pillar and breast plan of working, by which a larger portion of the pillar coal is recovered. It is not applicable to very thick seams nor can it be successfully employed in steep pitching beds.

The breasts and pillars are laid out in groups covering a considerable area; each group is surrounded by a thick pillar of coal which is broken only by the passages used as gangways and airways.

When a group of breasts is exhausted the work of withdrawing the pillars is begun at the extreme end of the panel. As the pillars are worked back and the pillar coal sent out of the mine, props are placed at short distances apart to prevent roof-falls near the face, but it is impossible by this means to prevent a general settling of the roof. When all the pillars have been drawn the props not already crushed are knocked out and the roof is allowed to settle and accommodate itself thoroughly to the new conditions before the pillars are withdrawn in the adjoining panel. After two adjacent panels have been exhausted, the coal contained in the large panel pillar may be obtained by robbing out the gangway and abandoning it, but if this gangway is needed for ventilation or future developments, the division or boundary pillars are left until that portion of the mine is to be exhausted. Several modifications of the panel system have been devised for anthracite mining. That invented by Col. D. P. Brown is so different from the panel system of Mr. Buddle* as above described, that it can hardly be called a modification of his plan.

Col. Brown has been experimenting with his new system for the last twelve years and is now of the opinion that it is best adopted to mining the Mammoth bed on pitches of from fifteen to thirty-five degrees and that it gives especially good results as contrasted with the pillar and breast system, where the coal is free, the top rotten and the mine gaseous.

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*Invented by Mr. John Buddle, Wallsend, Eng., about 1832.

The following resumé of Col. Brown's description of his system,* will show the great difference between this and Mr. Buddle's panel system.

When the Mammoth seam is pitching from fifteen to forty-five degrees and has its usual thickness in the Middle coal-field, of from thirty to fifty feet, the panel system may be commenced by driving the gangway and airway as in ordinary pillar and breast workings, but instead of opening breasts by driving up a shute every eight or ten yards along the course of the gangway, a shute is driven up through the seam to the gangway above, at a distance from the slope, shaft or airway sufficient to leave a thick pillar of coal to protect the main opening. (See Fig. 1, Atlas Plate No. XVII.)

At a distance of thirty or forty yards from this shute, a shute is opened, and at about thirteen yards from it a second shute is opened. These two shutes are driven up and opened out into two twin breasts, about eight yards wide, separated by a pillar of coal about five yards thick. A second pair of twin breasts is opened fifty or sixty yards from the first pair, and a third pair at an equal distance from the second, and so on. Midway between each pair of breasts a shute similar to the first is driven up to the gangway on the overlying lift.

These shutes are constructed so that the coal may run freely, and are provided with a travelling way on one side, giving the miners free access to the workings.

Small headings or gangways are driven in the bottom bench of coal, at right angles to these shutes, and about ten to twenty yards apart, the lowest heading being within six or eight yards of the gangway, as shown by the illustration on Atlas Plate No. XVII.

After the twin breasts are exhausted, the pillar between them is robbed out; but when the roof is poor, only a small portion of this pillar can be obtained.

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* See Report A^2, Second Geological Survey of Pennsylvania, page 33; Miners' Journal, Pottsville, January 20th, 1870; Report of the Mine Inspectors for 1873, page 187.

To begin mining the coal in the panel, (after the breasts are robbed out and abandoned) work is commenced in the upper heading or gangway by widening out the end like a breast, and extending the working up to the outcrop or to the next lift above, making the opening wide enough to bring down the top when a skip is taken off the solid. The coal thus falls down on the slope to a point under cover, where the laborer can load it in safety. A small mine car or buggy is used in the panel gangways or headings to transport the coal to the shute, down which it is thrown to be loaded in mine cars standing in the gangway below.

When the upper panel gangway has been worked back a short distance the next gangway below it is started, and this process is continued until the panel is exhausted, the miner always having a safe retreat into the panel gangway when a fall of coal or roof is imminent.

The full black arrows show the ventilation while advancing, the dotted arrows the ventilation when withdrawing. Col. Brown uses his fans for blowing instead of by the ordinary method of ventilation by suction, for which see the chapter on Ventilation.

When the pitch is from fifteen to twenty or twenty-five degrees, the central shutes may be given a greater inclination by slanting them in the bottom bench, and gradually rising through the coal to the top bench.

When the breasts are not worked up as far as the next lift above, a small airway is driven up to the return airway above, to carry off the foul air without returning it through the lower workings. New workings by this system have lately been opened on the fourth lift at the Shenandoah colliery—Lehigh colliery No. 3.

Colonel Brown advances the following as some of the advantages of this system, but states that at some of the collieries in the Second basin "the conditions are such that the old pillar and breast plan can be adopted as the most economical mode of mining, and with fair results as to the quantity taken out." The conditions to which this panel system are especially adapted have already been indicated.

"What it costs to open the panel and buggy coal to the shutes is compensated by the coal being more easy to mine, (from a loose end,) requiring less powder, timber, and labor, than working coal from a solid face.

"The cost of loading coal from the panel shute is less than a fourth of the cost of loading with a shovel, this item alone equalling about ten cents per ton of coal.

"The increased portion that can be taken out of the vein is an item that will amount to more than first cost of colliery, and will amply repay the operator for the additional cost of opening.

"If the system of panels is properly begun, and worked with a moderate degree of skill, *fully 80 per cent.* of the panel coal can be extracted, while a very large proportion of the refuse can be packed in the . . gob.

"This packing the refuse in the mine is a large saving in handling and preparation of coal, and is fully equal to ten cents per ton."

The object of opening twin breasts between the panels is to obtain a supply of coal while the panels are being prepared for working; they are, therefore, not an essential part of the system.

Boundary Plan.—A modification of the pillar and breast system, sometimes confounded with the panel system, and often improperly known by that name, is in use in many districts. It consists of separating the workings into districts by thick pillars of solid coal, in order to localize any movement of the roof and prevent a general squeeze or crush that would ruin the workings.

This plan has been used in some parts of the anthracite region, and in deep workings its adoption is certainly advisable. The actual method of working under this plan is exactly like any pillar and breast working, and, therefore, while it does not deserve recognition as a distinct system, it may be appropriately designated as the Boundary or Barrier plan.

Veith's Boundary Plan.

The plan devised by Mr. John Veith, Mine Inspector for the Philadelphia and Reading Coal and Iron company, accomplishes more than is attempted by the ordinary barrier plan. By leaving a thick stump of from fifteen to twenty yards between the gangway and the breasts, he is enabled to take out a larger percentage of the pillar coal; and by leaving barrier pillars fifty yards wide at intervals of about every two hundred yards along the gangway, the workings are divided into panels, so that, when the roof begins to move on the removal of the piller coal in the panel, the "squeeze" or "crush" is limited to that group of breasts. The gangway stumps and barrier pillars are not robbed out until the mine, or a portion of it, is to be abandoned.

A somewhat similar plan has been in use at several other localities for a number of years. This plan is to leave every eighth or tenth breast unworked, thus leaving a pillar of untouched coal from twenty to twenty-eight yards thick, which is not worked until reached in the process of robbing back.

4. "Square-work." *

The "square work" method of mining the thirty foot ("10-yard" or "Dudley Thick") bed of South Staffordshire (Eng.) is not adapted to steep dips, and even under the most favorable circumstances the risk to life and the amount of coal left in the workings by this plan are probably greater than in well managed pillar and breast workings.

The workings are opened from the gangway in the form of a square or rectangle, fifty yards or more in size. This "square of work" is separated from the gangway by a pillar of coal six to ten yards thick, which is broken only by the narrow entrance from the gangway, termed a "bolt-hole." Workings are opened out in this area by driving in the lower benches, opening out a series of parallel rooms five to ten yards wide, and a second series crossing the first at right angles. By this method a series of pillars about eight or ten yards square is left to support the roof in each "square of work."*

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*See "Coal and Coal Mining," by W. W. Smyth, London, 1875: also Transactions North of England Institute of Mining Engineers.

The top benches of the coal are reached either by standing on the loose coal or by ladders or scaffolding. Only a very small percentage of the coal left in these square pillars can be recovered, and the risk incurred by the miner in attempting to rob them, must always be very great, while the difficulty of maintaining efficient ventilation under such circumstances will act as an effectual bar to the adoption of this system in those portions of the anthracite fields in which it might otherwise be adopted. This system is in all its essential characteristics a species of panel work and should be so classed.

Anthracite Mining Methods.

As already indicated in the preceding pages, the pillar and breast system is the one by which almost all anthracite mines are worked, but the details of the system under different mining conditions vary so widely that a comparison of mine maps from different districts, or a hasty trip through collieries in different localities gives the impression that the coal is mined by several systems having nothing in common; but a close inspection discloses the fact that they all have three identical features; 1st. The breasts, or rooms, or working places are all long and comparatively narrow; 2d. They are driven nearly parallel to each other, and 3d. They are separated by long narrow pillars of coal, broken only by small openings for ventilation.

These features common to all anthracite workings (except Col. Brown's panel system) are the identifying characteristics of the "pillar-and-breast," "pillar-and-room," "post-and-stall," "stoop-and-room," or "bord-and-pillar" system as described in the preceding pages.

A perusal of the introductory chapter in which the geological peculiarities of the anthracite coal beds are described in some detail, will sufficiently explain why the mining method in some districts differs so radically in its details from that employed in others.

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* Known also as "sides of work."

At present, beds from three and a half or four feet up to sixty or seventy feet or more in thickness are worked,* and on dips varying from horizontal to vertical, and it is a somewhat remarkable fact that the same general system can be employed in mining the coal from a six foot bed lying nearly horizontal, that is adopted in working a fifty foot bed standing nearly vertical; but the details of such workings differ radically.

The plans in use at present may be conveniently divided into two classes; thus:

1. Workings on beds dipping from zero, or horizontal, to an angle of about twenty-five degrees.
2. Workings on beds dipping from about twenty-five degrees to ninety, or vertical.

These two classes may appropriately be termed "Flat workings" and "Pitching" or "Steep workings."

Flat Workings.

When the seam is flat it is usually opened by a shaft sunk through the overlying measures. Gangways are driven out from the bottom of the shaft, and when these have reached a sufficient distance,—allowing a thick pillar of coal to remain to protect the shaft,—breasts are opened from the upper side of the gangway. Ventilation is supplied by an airway driven parallel to the gangway and separated from it by from five to ten yards of solid coal, broken only by the necessary cross-headings. This airway is usually driven in the upper benches of the bed, so that it is somewhat higher than the gangway level, and, in very thick seams, is at times almost vertically above the gangway.‡

When the coal is quite flat the breasts are turned† at right angles to the gangway, but when the dip is too steep to

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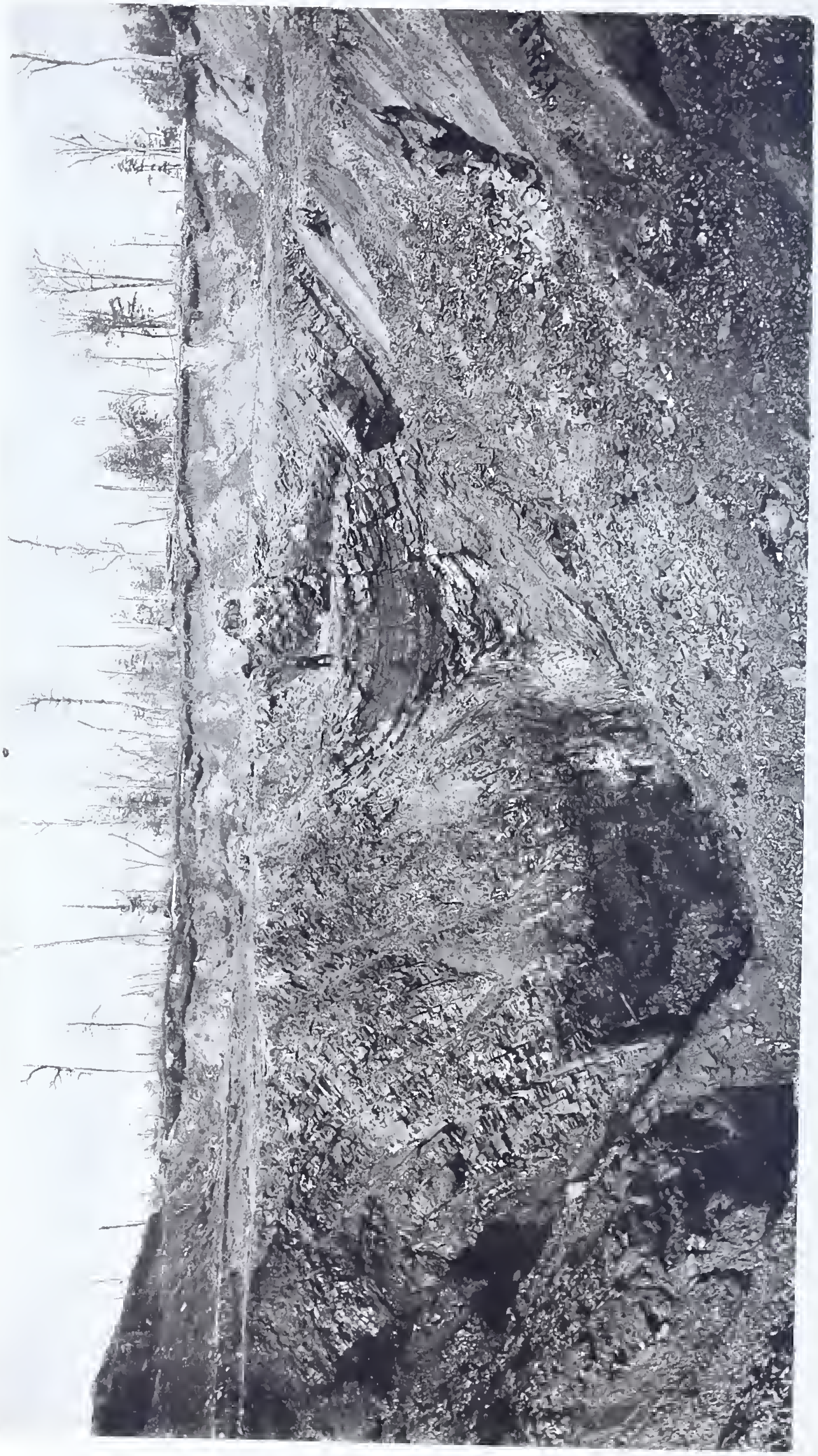
*At some places where the Mammoth bed is folded over on itself, thus doubling its thickness, more than one hundred feet of coal have been obtained in a single working.

† Opened.

‡ This statement applies only to steep dips.



STRIPPING AT HOLLYWOOD COLLIERY NO. 2,-- LOOKING EAST. (PICTURE REVERSED.)

ARTOTYPE E. BIERSTADT N. Y.

allow a mine car to be used in the breast if so driven, (from 3° to 7°,) it is opened at an angle to the gangway, thus decreasing the inclination of the floor; as shown by Fig. 2 of Atlas sheet No. XXI.

Two plans of opening such breasts are in common use: in one the breast is opened at the gangway to its full width of from eight to twelve yards, in the other an opening just wide enough for the mine car is driven in for eight or ten yards, and the breast is then opened out to its full width.

When the dip is very flat the breasts are sometimes driven a long distance, as in some mines of the Wilkes-Barre district, where breasts over two hundred yards long are not uncommon.

The inclination of the bed usually limits the breasts to three, four, or five hundred feet in length, and coal lying at a greater distance from the gangway is mined from a second series of breasts opened from a second gangway driven above the first series, or driven directly across its upper edge,—called a "counter-gangway."

On steeper dips the mine cars from this gangway are lowered on a self-acting plane to the shaft level gangway.

When the coal lying above the level of the shaft is exhausted, coal lying at a lower level is opened by an inside slope and mined in a similar way.

When the dip of the coal is from five or six to twelve degrees, the plan of working is the same; but as the mine cars cannot be taken up into the breasts, the coal is brought to the gangway in a small car called a "buggy," holding from half a ton to a ton of coal, as shown by Fig. 4 on Atlas Sheet No. XXII.

When the inclination of the seam exceeds twelve degrees the buggy may be replaced by a sheet iron shute; and from this inclination up to twenty-five degrees of dip the general plan of working is the same.

In the chapter on gangway driving the grade usually adopted is stated as ranging from three or four to twelve inches per hundred feet, but in working on beds dipping at a very slight angle the grade is often very much steeper, and this is especially true of the gangways driven off at an angle to the main haulage road to furnish an outlet to a set of *counter gangway* workings. That portion of such a gangway running diagonally across a series of breasts is called a "*run*" or "*the run*," from the fact that the loaded cars run down by gravity.

The miners commonly designate roads of different grades by the number of wheels that must be spragged to properly control the descent of the car,—thus a "*one*," "*two*," "*three*" or "*four-sprag-road*" means a road on which one, two, three, or four wheels of the car must be spragged, or, as one sprag is required to sprag each wheel, a road on which one, two, three or four sprags are needed for each wagon.

In flat workings the necessity of driving many branch (counter) gangways, and the very irregular course of the main gangways and also of the breasts, give the workings, as shown on a mine map, an appearance of having been opened without reference to any systematic mining method.

At first sight some of these maps resemble not a little maps of some of the large European cities in which the course of the streets has been determined by no preconceived plan,—but a closer inspection always discloses a system regular and simple in its conception, but modified and complicated by the presence of anticlinal rolls, small basins, "hog-backs" or faults.

When the dip of the coal is steep and the beds crop out at the surface, the usual method of development is by slopes, and the method of working is known as "lift mining."

Workings precisely similar to those operated through a slope are common at shaft collieries, but under such circumstances the coal is usually lowered from the higher levels to the shaft-level gangway by means of self-acting planes, hence the name "lift mining" applied to such workings is a misnomer. When, however, coal lying beneath the shaft-level gangway is developed by inside slopes, the method of developing, working, and handling the coal is identical with that at slope collieries.

When the mouth of the slope is located at a considerable elevation above the nearest water course, a gangway is usually opened out as soon as the slope has reached a depth corresponding to the lowest point on the outcrop of the bed, and driven through to daylight, (usually in a ravine,) thus forming an open water-level drift, commonly known as the "water-level gangway" or "water-level lift." Instead of starting this gangway from the slope it is often driven in from the surface.

When no point on the outcrop is low enough to furnish an outlet to the water-level gangway a tunnel is frequently driven in across the intervening measures.

At a distance of from seventy-five to one hundred and twenty-five yards below the water-level gangway,—if there is no water-level gangway then below the mouth of the slope,—a gangway is started on each side of the slope to open up the "First Lift,"* and at a corresponding distance (generally a few yards less) further down the slope another gangway is driven to open up the second lift of workings.

When the dip is very steep, and especially when the coal is loose enough to run, the second-lift gangway is driven and breasts are turned ready for mining while the first lift is being worked, but very little actual mining is done until all the coal that can be obtained is taken from the first lift, and the gangway and air-courses at that level are no longer needed. The same method is followed in opening and working lower lifts.

On more gentle dips, and especially in hard unbroken coal that has no tendency to "run," two or three lifts are frequently worked at the same time. (See foot-note, p. 134.)

The distance to which the gangways are driven on each side of the slope, or in other words, the lineal distance worked from a single opening is limited by the cost of keeping the gangways open, and by cost of haulage to the

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*At some collieries the water-level lift is called the First Lift and the lift below the Second Lift; this difference in naming the lifts arises from the fact that when the coal from the water-level drift is taken out *through the slope*, the *coal* is lifted, and the miner therefore calls this the first lift; forgetting, or in ignorance of the fact, that the term has special reference to the lifts of water, or number of pump lifts. In the anthracite region this term has lost its significance in this respect, and is understood to embrace *all the developments* opened up from a gangway at certain level. Thus at many collieries the second lift of pumps is found on the "Third Lift," etc. The term "lift" is virtually used synonymously with the term "level."

outlet, and this applies as well to shaft as to slope workings.

When the coal is hard, the gangway roof good, and the ventilation and mine conditions such that the gangway timbering will stand a number of years without requiring renewal, or when very little timber is required, it is often cheaper to mine coal lying two, three, or even four miles from the slope and haul it through this gangway than to make a new opening; but when the coal is soft, the roof poor, and heavy timbering necessary to keep the gangways open, it may be cheaper to open a new slope than to attempt keeping even one mile or less of gangway open.

The haulage charge is greatly lessened on long hauls by the use of locomotives, but the heat and waste gases from these engines, amount of good air they consume, and the presence of fire-damp, prevent their use at many collieries.

In opening up a new lift the breasts are turned as fast as the gangway is driven, and mining commences at once, but the first breasts are not opened until the gangway has reached a sufficient distance from the slope to leave a good thick "slope pillar" of solid coal, as shown by Fig. 1 on Atlas sheet No. XXI.

To insure the maximum amount of coal from a given working, the gangway should be driven to the limit of the proposed working before commencing to mine the coal. Mining should then be commenced by opening the first breasts at the limit and working back, robbing the pillars as soon as the breasts are exhausted, and following this up by robbing out the gangway; but as this plan† entails a heavy outlay in dead-work from which the returns are not immediate, it has not been adopted. (See Chap. XXVIII.)

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* Run: to break loose and slide down the dip into the shutes below without mining—to flow spontaneously. When a breast is opened in a coal bed standing at a high angle of dip and fissured throughout by cleavage joints and lines of fracture, the coal often begins to break away from the solid without assistance from the miner, when the coal is said to "run." Breasts on the second and third lift will sometimes continue to run until portions of the surface at the outcrop are drawn down and loaded with the coal in the mine cars.

† Which might be appropriately termed "Pillar-and-Breast-withdrawing" and the common plan "Pillar-and-Breast-advancing."

The breasts are not worked through into the gangway above, but are driven up to within five, ten, or fifteen yards of the overlying gangway. The pillar thus left on the lower side of each gangway is called the "chain-pillar."

Chain-pillars are left to insure the safety of the gangway; but often not so much because there is any necessity for keeping it open as a gangway as because it is useful as a water-way, catching a large quantity of water that would otherwise find its way into the lower workings; but in thick seams the lower gangways are robbed out and the chain-pillar holed through from below as soon as the gangway is no longer needed as a haulage road or airway, as it is seldom good policy to attempt to keep any of these courses open (except the water-level gangway) as water-ways.

When the breasts on any lift are worked out the pillars are robbed by taking off from each as thick a slice as possible, and then, if the gangway is to be abandoned, the stumps and gangway roof coal are robbed out. In thick seams this is extremely dangerous work, but in seams of moderate thickness, with fairly good roof, a large percentage of the stump and gangway coal can be obtained with little risk to the miner.

When the coal runs, it is impossible to hold the water on the upper gangways, as each gangway, with the gangway stumps, will draw down into the workings on the underlying lift.

The distance between lifts is always reckoned at so many yards measured on the slope of the bed. As all the timbering used in the breasts for manways, props, cross-headings, etc., as well as tools, powder, and other supplies, must be raised by the miner from the gangway below, the length of a pitching breast is necessarily limited to the distance which these supplies can be raised at a certain cost. On steep dips the slope distance between lifts averages from seventy-five to one hundred yards,*—in flat workings the breasts are driven a much greater distance.

It often happens that the distance between the two lifts is greatly increased by a flattening of the dip, making the length of the breasts too great for economic working. Thus if a mine is opened by slope on a bed dipping sixty degrees, and the distance between the lifts is ninety yards, and in driving the gangways away from the slope the dip is found to diminish to thirty degrees, the distance between the two lifts is increased to about one hundred and fifty-six yards.

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*When little timbering is needed this distance is sometimes increased to one hundred and twenty or thirty yards.

Under such circumstances a second gangway (termed a "counter gangway") is driven about midway between the two lifts, and a subordinate series of breasts opened to work the coal lying between it and the upper level. The coal is either lowered from the counter gangway to the gangway below by a self-acting plane, or is allowed to slide down a shute,—known as a counter shute,—from which it is loaded into mine cars on the lower level.

As counter gangways often have no direct connection with the slope, all the timber and other mine supplies, rails, mine cars, etc., necessary in developing and operating the gangway and breasts, must at first be raised from the lower gangway by hand, but after the workings have been holed through into the upper gangway, these materials are lowered through a breast or shute used especially for that purpose. When the counter-gangway workings are extensive, the supplies are lowered on a self-acting plane, built in the shute or breast used for that purpose, otherwise they are allowed to slide down by gravity on the floor of the shute; machinery and mine cars are sent down or brought up in small pieces to be fitted together afterwards. They are either lowered by hand, raised by windlass, or skidded down on mine timber.

To avoid the necessity of raising coal from two different levels, the first or second-lift gangway is sometimes operated as a counter gangway, as shown by Fig. 1, Atlas sheet XXI.

When the coal runs freely without mining, the length of the breasts is not limited by the expense entailed by raising timber and other supplies,—as no work is done in the breast after it is opened out,—and one lift might be made the whole length of the slope from outcrop to the bottom of the basin and all the coal taken out from a gangway driven at the bottom, were it not that the phenomena of run mining present some features fatal to this plan.

After a series of breasts have been opened out, and the coal begins to run, the breasts rarely maintain their regular width and shape.

1. In some cases the breast gradually becomes more and more narrow until at last the coal ceases to run, in which case the miner must enter the working and widen it out until it again begins to run.

2. The coal sometimes runs much more freely, and instead of the breast narrowing, it widens out, encroaching on the pillars, until two adjacent breasts may be united by the entire destruction of the pillar, following which extensive roof-falls occur that may effectually close the workings, thus locking up a large amount of coal between the roof-fall and the overlying gangway.

3. The coal may run irregularly from either side of the breast, so that the central line of the breast is not vertically up the dip of the coal, but at a considerable angle to it, depending upon the direction of the cleavage joints, the cleat, or upon the existence of lines of fracture or local variations in the physical condition of the bed, resulting from the pressure to which it has been subjected.

The distance between lifts, or the length of the breasts, in free running coal, is therefore limited by the risk of losing large masses of coal by roof-falls locking up the workings.

Pillars.

Pillars are designed to accomplish two objects:

1. To prevent the roof-rock from falling in and prematurely closing the breasts, (and injuring the miners,) and,

2. To sustain the enormous weight of the superincumbent mass of rocks and prevent a general movement of the strata. Such a movement is known as a "crush" or "squeeze," from the fact that it crushes down the pillars, not those between the breasts only, but the gangway stumps and chain pillars, destroying at once the workings, the gangways, and the airways.

To accomplish this last object, it is of course necessary to leave much thicker pillars in deep than in shallow workings. When the roof is poor and soon falls into and closes up the breasts, a considerable portion of the superincumbent weight is sustained by this débris; but when the roof is firm the pillars bear all of this weight,—hence to avoid a crush or squeeze larger pillars are needed when the roof is good than when it is poor.

To accomplish the first object, the pillars are closer together (but not so thick) when the roof is poor than when it is strong and firm.

The great waste in mining our thick anthracite beds may be attributed principally to the necessity of leaving large pillars, stumps, and chain-pillars, to prevent closure of the workings by crushes and extensive roof-falls.

Even before any attempt is made to rob the pillars, crushes not infrequently occur destroying large areas of partly worked coal, and robbing can seldom be carried to any extent without inducing a squeeze that may stop all further operations in the area affected by it.

When, as in some portions of the anthracite fields, a large bed of coal is underlaid at a short distance by a seam of moderate thickness, the two may be worked by an ingenious plan which has lately been adopted by the P. & R. C. & I. Co., and which, in the absence of a better term, I shall designate as "Rock-shute Mining."

Rock-shute Mining.

This plan was especially devised for mining the Mammoth bed, but is equally applicable to any other thick seam when underlaid at a moderate distance by a thin bed of coal.

It was first proposed as a method by which the coal in large pillars and that imprisoned by extensive roof-falls and crushes, might be recovered; but it might under some circumstances be advantageously employed in working coal that has not yet been touched.

The plan may be very briefly described; as it is a comparatively new departure, its details will doubtless be improved after its defects, if any, have been discovered by practical working tests.

The method, as at present being experimented with by the Philadelphia and Reading Coal and Iron company at the Plank Ridge and Knickerbocker collieries, consists in opening the gangways and airways in the underlying seam (the Skidmore), and then tapping the Mammoth bed by rock shutes. It will be employed at present to recover the pillar coal, the top benches not mined, and the coal imprisoned by roof-falls and localized crushes in areas already worked.

The shutes may be driven from four and a half to seven feet high, and seven to twelve feet wide, at an angle at which the coal will slide by gravity to the gangway below.

Similar work has been done in the past on a small scale at a number of localities, but the plan has never been systematically tested by extensive workings. A somewhat similar plan is shown by Fig. 2 on Atlas Sheet No. XXIII, (which will be described in the chapter on the methods of working breasts,) but in this case gangways are driven in the upper bed as well as in the lower seam, hence the amount of coal recovered is not appreciably increased.* The Lehigh Coal and Navigation company has also employed this plan to recover masses of coal imprisoned by large roof-falls, and the same plan has been used in other localities; but none of these operations can be considered as representing this plan as a mining method distinct from the ordinary pillar and breast working. The lower seam was opened and worked in the ordinary way to mine the coal contained in it, and when the workings extended beneath large pillars or imprisoned coal in the overlying bed, rock shutes were driven through to recover it.

The plan proposed above, and to which I have given the name "Rock-shute Mining," contemplates a very different sequence of operations, which may be summarized thus:

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*The effect of this plan was not so much to reduce the mine waste as to cheapen the cost of mining, (see the following chapter,) by reducing the amount of dead work.

1. The opening of all gangways and airways in the lower seam to develop coal as yet untouched in a thick seam lying a few feet above it.

2. Developing the thick bed by a regular series of rock shutes driven up from the gangway below; workings being opened out from the shutes as in ordinary pillar and breast working.*

3. Driving the breasts to the limit of the lift and robbing out the pillars from a group of breasts as soon as possible, even if a localized crush is induced.

4. After one group of breasts is robbed out and the roof has settled, opening a second series of shutes for the recovery of coal from any large pillars that were not yet robbed when the crush closed the workings.

5. While the work of recovering the pillar coal is in progress a second group of breasts may be worked, and the process continued until all of the area to be worked from that gangway has been exhausted. The same process is employed in opening lower lifts.

6. When all of the upper bed coal has been exhausted, the lower seam may be worked by the ordinary pillar and breast plan. Workings in this seam may be carried on *pari passu* with the upper bed, but to avoid the possibility of a squeeze destroying these workings, very large pillars must be left. After exhausting the upper workings these pillars may be advantageously worked by opening one or two breasts in the center of each, and when these are worked to the upper limit, attacking the thin rib on each side, commencing at the top of the breast and drawing back.

When the roof of the lower bed is good, the cost of timbering and keeping open the gangways and airways will be considerably less than if these were driven in the upper seam, and this difference in some cases may be sufficient to pay for driving all the rock shutes.

The increased percentage of coal obtained from the upper bed, and the fact that the lower coal is opened for mining with no further cost for gangway and airway driving and timbering, combine to make this plan worthy of careful consideration by every mining engineer, colliery proprietor, and coal land owner.

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* The panel system or some other plan may be found better than pillar and breast workings.

Some of the problems yet to be solved before it can be recognized as an efficient mining method are of considerable importance. Prominent among them (all of which must be solved by practical trials,) are the following:

1. The maximum distance between the two beds, or the length of rock shute that can be driven with satisfactory financial results.

2. The maximum dip on which such workings can be successfully opened.

3. The minimum thickness of the upper and also of the lower seam which will yield results warranting the additional outlay when the rock shutes are of considerable length.

In the Knickerbocker and Plank Ridge collieries the Skidmore bed underlies the Mammoth seam by from fifteen to twenty feet. The Skidmore bed varies from four to six feet in this locality.

In various portions of the anthracite regions large areas of the Mammoth bed were worked twenty or thirty years ago by simply mining the lower benches (sometimes the second bench) but these old workings still containing the upper benches of coal and much pillar coal have long since been closed by the weight of the overlying strata. At some of these localities the conditions are favorable to reworking by the rock-shute method, and by it a large percentage of coal left in these old workings can doubtless be recovered.

The *origin* of this plan was probably accidental; its *amplification*, and *adaptation* to thick seam mining, is due to the ability and progressive energy of some of the mining experts of the anthracite region, but to whom, whether to one or to several, I have not been able discover. The occurrence of split seams or twin beds separated by a few feet of slate or rock, as shown by Fig. 2, Atlas plate No. XXIII, probably first suggested the very natural expedient of making the lower gangway do double duty by taking out the upper coal through rock shutes and sending it out of the mine through the lower bed gangway. From this plan to that above described as the rock-shute method the transgression is by easy steps, and it needed only an inventive or rather adaptive mind to see how well suited it is to thick bed workings.

Under conditions favorable to the successful working of this method, the amount of coal mined from the Mammoth bed could probably be increased from the present yield of say fifty-five per cent. to seventy or eighty per cent.

It has been claimed that the actual cost of opening the Mammoth bed by this plan in the Mahanoy and Shenandoah districts, would be less than by the ordinary plan ; the assumption being that the cost of all dead-work in the Skidmore bed would be realized from coal mined from that bed, and that the only extra charge to be made against the Mammoth developments for dead-work, would be the cost of driving the rock shutes. Whether this claim can be substantiated by practical tests, remains to be proved. In gaseous workings a considerable additional expense would doubtless accrue from the necessity of driving independent airways through the intervening rock.

It has often been remarked, especially by experts not engaged in practical mining operations, that while the mechanical appliances of coal mining, the winding, pumping, and ventilating machinery, have been most remarkably improved in the last thirty or forty years, the improvements in mining methods are by comparison insignificant.

That such is the case must be admitted,—it is at least partly true,—but it may easily be explained. In the mechanical branches of mining, the mining engineer has had at his disposal the improvements in machine construction effected by the united labors of mechanical engineers throughout the civilized world, and he has not been slow in adopting and adapting to the peculiar requirements of mine work, improvements in steam engineering, pumping machinery, and miscellaneous mechanical inventions. The number of engineers and inventors engaged in mechanical work is very large, the number of mining engineers comparatively small, and these latter are more widely separated, and their incentives to invention not so great. Again, as there is always a large amount of machinery to be designed, erected, and kept in repair at extensive workings, a mechanical engineer is frequently chosen to superintend the mining work, and without knowing or caring much about mining, he is placed in the position that should be occupied by an expert mining engineer.

If our mining methods are to be improved in like ratio with the mechanical appliances of mining, these two professions must be divorced. The mechanical engineering must be done by mechanical engineers, and the mining operations superintended by mining engineers, and by this division of labor only can the best results be obtained.

The collieries of some of the largest anthracite mining companies are now managed in this way, but the practice has not yet become general.

While a certain knowledge of machinery and mechanical appliances is absolutely necessary to the mining engineer, it is not necessary or even advisable, that he be a mechanical engineer; for as mechanical engineers, as a class, are more interested in machinery than in mining methods, there is small hope of improvement from them, in any other than the mechanical appliances of mining.

The mining engineer is also too frequently required to perform all the duties of a business superintendent, and these duties, (many of which are entirely foreign to his duties as mine superintendent and engineer, such as supervision of all accounts, pay-rolls, shipment of coal, payment of bills for supplies, miscellaneous correspondence, etc., etc.,) occupy so much time that he has little, if any, leisure for the consideration of improvements of any kind.

These are some of the apparent reasons why mining as a science has not advanced step-by-step with the mechanical arts, and the condition of the science in countries other than our own, points to the operation of the same or similar causes.

It is especially in the mining of thick seams such as the Mammoth bed, that mining appears to have been almost at a standstill for the past two decades, and not a few professional mining men believe that no further advance is possible in this direction, at least for many years to come. In mining the thick ten-yard coal of South Staffordshire, Eng., the percentage of waste is about equal to our waste in Mammoth bed mining; from this it appears that in thick bed mining we can learn little from English engineers.

By some radical change only in the present method of working, can any marked improvement be effected in thick coal mining. What this change shall be, the future must decide, but the need of a change of some kind is now thoroughly appreciated by all anthracite miners, and that a better method will eventually replace the present wasteful system cannot be doubted.

Modifications of the details of working and opening breasts are constantly being made, but from these the improvement must necessarily be small,—to accomplish any great reduction in the percentage of coal left in the ground, the whole system of mining must be radically changed.

This subject will be considered in detail in the Chapter on Waste. (Chapter XXVIII.)

CHAPTER IX.

Methods of Opening and Working Breasts.

The places from which the coal is mined are known throughout the anthracite coal-fields by the term *breasts*, but in some localities they are also often called *rooms* or *chambers*, these latter terms being most commonly applied to flat workings.

In order to describe more clearly the different methods now in actual use in the anthracite coal fields of opening, ventilating, and working breasts, they will here be considered as divisible into three classes, according to the dip of the coal, thus:

1. Wagon breasts; in coal dipping less than seven degrees;
2. Buggy breasts; in coal dipping from six or seven to ten or twelve degrees;
3. Shute breasts,—Pitching breasts; in coal dipping from twelve or fifteen degrees to ninety degrees or vertically.

Wagon breasts, or those into which the mine car is taken on a track laid from the gangway to the working face, admit of few variations in the method of working, and this is also true of buggy breasts; but there are several methods of opening and working in steep-pitching coal which differ widely from each other, and these differences result not only from the possibility, but also from the necessity of varying the method to meet the peculiar requirements arising from unlike conditions.

The seams now being mined are the best and thickest in the region, (the smaller seams are not worked on a large scale, nor to any extent below water-level,) and there are few large workings on seams less than six or seven feet thick. It is therefore unnecessary to here describe the operations in beds of less size; and the following details will be considered as applying to beds from six up to sixty or seventy feet in thickness.

As already stated in the introductory chapter, the coal seams contain a considerable amount of interbedded slate and "bone," and in some beds the amount of this refuse is so great that the bed cannot be profitably worked unless this material can be left in the mine.

Differences in the dip, in the thicknesses of the beds, in the character of the coal, in the relative amount of refuse contained in the different beds, in the character of the roof, and in the amount of gas; are the variable factors that necessitate the adoption of several widely different methods of opening, ventilating, and working pitching breasts.

1. *Wagon breasts.*

Wagon breasts may be opened at once to their full width on the gangway, but to insure the safety of the gangway roof without resorting to heavy timbering, they are commonly opened by a narrow shute* or wagon-way, driven in for a distance of from five to ten, or in some cases fifteen yards, and then opened out to the proposed width, thus leaving between the gangway and the breast a square pillar of coal commonly called a "stump" or "gangway stump."

When the coal is very flat, not dipping more than two or three degrees, the breasts are turned at right angles to the gangway, but when the dip is from four to six or seven degrees they are turned at an angle to the gangway, as shown by Fig. 2 on Atlas sheet No. XXI. The heading running across the lower end of the breasts is the air-course which is driven as a continuous channel to supply ventilation during the gangway driving.

In workings of this class it is rarely possible to drive the breasts in a straight line and at the same time obtain a favorable grade for the mine cars, and in flat workings like those of the Wilkes-Barre district, the breasts are frequently curved or bent at a sharp angle to conform to the contour of the bed, or to an abrupt change in the direction of dip.

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* A misnomer when applied to flat workings of this class.

The section accompanying the above illustration shows this method of working as applied to a bed about eight feet thick in which the roof is rather poor and needs propping; the breasts are driven ten yards wide, the pillars are seven yards thick, and the gangway stump is ten yards deep.

As the breasts are driven in, ventilation is obtained by cross-headings driven through the pillar from breast to breast at distances varying from fifteen to twenty yards or more, this distance being governed by the amount of gas given off by the coal.

After the coal is mined it is shovelled into the mine car, the miner being careful to exclude from it all large lumps of slate, "sulphur balls," and rock, which are thrown on one side of the working and constitute what is known as the "gob."*

When the gangway has a considerable grade, an advantage is obtained by inclining the breasts backward towards the outlet, as shown by the illustration (Fig. 2, atlas sheet No. XXI) instead of away from it.

The breasts usually have sufficient grade to allow the loaded mine car to run out to the gangway by gravity.

When the roof is poor the breasts are not driven as wide as when working under a strong roof, but the variation in width is comparatively small. Breasts are rarely driven less than eight yards wide, even when the roof is very poor, and a width of about twelve yards is the maximum.

The coal is more readily loaded into the mine car when the breast road is laid up the center of the working, but when the roof is very bad, greater safety, with less propping, is secured by laying the road close to the rib.†

Pillars only five or six yards thick are not uncommon in some of the Wyoming valley mines, but for deep workings this width is not sufficient to insure immunity from serious crushes; and this is especially true when a considerable lapse of time must occur before robbing is commenced. In deep workings pillars are now left as wide, or even wider, than the breasts.

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*English Goaf.

†"Rib,"—the side of the pillar.

The system of ventilation adopted, and the amount of gas given off by the coal, govern the size of the cross-headings. They are often driven as small as possible, but when the gas is troublesome, and the air current is coursed through a group of several breasts, they are made very large. About four feet by five or six is as small as they are commonly driven, and a size of six by eight feet, is about the maximum, except in very gaseous workings.

When the bed is less than ten (or twelve) feet thick, it is supported by propping (when necessary), in thicker seams the cost of long heavy props and the time and labor spent in setting them are so great an expense that no attempt is made to support the roof.

Two methods of working flat breasts in thick seams are in use. In the first method the breast is opened out and driven to the limit in the lower bench of coal, say to a height of from five to eight or ten feet, the top, if necessary, being supported by props, and the top benches are blown down afterwards, beginning at the face and working back; or when the roof is good and there is no danger of a roof fall closing the working, the upper benches may be worked in the opposite direction, beginning at the wagon-way and driving out towards the limit.

In the second method the working is opened at once to the full height of the seam, and so driven to the limit of working.

When the roof is very poor and the coal itself is hard enough to make a fair roof, or contains a parting of slate that will make a good roof to the lower opening, the first method is by far the better plan of working, and if the upper benches are worked back towards the outlet the danger of closure from roof-falls is avoided.

It frequently happens that the coal is of such a character that, after being undermined by the removal of its lower bench, the upper benches need only a few shots to start them and then fall in large masses that must be broken up with powder.

When the full height of the bed is worked at one operation, the miner usually first underholes the top by mining the lower bench to a depth of a few yards, varying with the thickness of the bed, and then blows down the upper benches by a few shots located near the top. In very soft coal the top benches often begin to fall as soon as they are undermined a distance of two or three yards.

The miner reaches the top benches either by standing on the fallen coal, by a rude timber staging or by ladders.

2. *Buggy breasts.*

When the dip of the coal exceeds six or seven degrees, the grade is too great to take the mine cars into the working places*, and when the dip is less than twelve or fifteen degrees, as the coal will not slide down to the outlet, it is transported from the face to the gangway in small mine cars or wagons known as "buggies," and the coal is said to be "buggied," the breasts being known as "buggy breasts."

Buggies vary in capacity from half a ton to a ton, and as they must be pushed up to the face on a track that has quite a considerable grade, they are made as light as is consistent with proper strength. They are provided with a door at the rear end, through which the coal is dumped on the platform or into the loading shute, and run on a track laid through the center of the breast.

On steep pitches wooden rails are preferred to iron, because the friction between the wheels and a wooden tram-road is greater than when iron rails are used, and the buggy is more thoroughly under the control of the miner, but on more gentle pitches iron rails are used. The guage is commonly from two feet and six inches to three feet.

A buggy breast as opened in a bed nine or ten feet thick is shown by Fig. 4, Atlas Sheet No. XXII, in which the buggy is run out on a staging at the lower end of the breast, so that the loading platform and shute may be made to hold a considerable quantity of coal.

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* Wagon breasts are occasionally opened in coal steeper than this, but the work of pulling the mine car up to the face is very laborious, especially when a large heavy car is used, and this practice is adopted by very few mining engineers. Wagon breasts have been opened on dips so great that a block and tackle were necessary to draw the car up to the face, but this practice seems to me to partake of the nature of a "lazy man's load."

When the bed is very thick the breast or shute may be opened near the top of the gangway and driven in horizontally until the floor is reached; by opening out in the top benches, sufficient room may be obtained to load the coal directly into the mine car over an apron projecting into the gangway. The expense of shoveling the coal from the platform into the mine car is saved by this plan; but it can seldom be adopted, and is objectionable from the fact that a large amount of bottom coal is left at the lower end of the breast that must be mined after the breast is exhausted, and if the bed contains much refuse this coal will be covered by a large amount of gob that must be removed before the coal can be taken up.

Buggy breasts are opened with a single shute, and in other respects are worked in the same way as wagon breasts, the coal being cleaned in the breast and the gob thrown on one side of the road bed.

For mining coal from buggy and wagon breasts the miner is paid a certain sum for each mine car of properly cleaned coal sent from the mine; the price varies at different collieries according to the size of the car, the thickness and character of the coal, and the amount of refuse contained in the bed.

Fig. 4, Atlas Sheet No. XXII, shows a buggy breast with the shute opened directly from the gangway, and a mine car standing in front of the platform in position for loading. A better plan is to widen out the mouth of the shute at the gangway and thus obtain room for a short siding, (just long enough to hold one mine car,) on which the car may stand while being loaded or until the miner is ready to load it. By this plan the gangway is kept open as a haulage road; when the other method is employed it is always obstructed by cars being loaded. In pitching breasts the cars are quickly loaded from a shute and the obstruction is only momentary; but when the coal is shovelled into the car by hand, the loading occupies from fifteen minutes to three quarters of an hour, in addition to any delay occasioned by the miners not being ready to load. On gangways used as haulage roads for a considerable number of breasts, the cost of the short sidings is small, in comparison with the time lost by frequent blockades on the gangway.

Such an arrangement is also resorted to in fiery mines, when for the purpose of more thorough ventilation, it is necessary to cut off all communication between the breast and the gangway, by laying the siding into the shute a sufficient distance to admit the mine wagon, and closing the shute by a door, which is opened only when a loaded car is taken out or an empty one taken in to be loaded.

3. *Pitching or Shute-breasts.*

In steep-pitching breasts the coal slides down through the breast to the shute, and through the shute into the wagon, by gravity. When the dip is from twelve or fifteen up to twenty-five or thirty degrees, sheet iron is laid on the floor of the breast and also in the shute to facilitate the movement of the coal, but on a dip of less than eighteen or twenty degrees the coal will not move freely even on sheet iron, and must be pushed down by the miner.

When the inclination is less than twenty-five or thirty degrees the workings may be opened and worked like buggy breasts, with one shute which ends in an apron projecting into the gangway high enough to allow a mine car to pass beneath it; but the coal instead of being buggied to the shute, slides down over sheet iron laid on the floor of the breast.

When this method is employed the gob is thrown on either side of the breast out of the way of the sliding coal. If the pillars are to be robbed by skipping one rib only, it is well to keep most of the gob on one side of the breast.

When the roof is good and the breasts are driven ten or twelve yards wide, they are frequently opened with two shutes, but unless this is necessitated by the method of ventilation, no great advantage is to be derived from it and the cost of opening is considerably greater.

When the dip exceeds thirty degrees the coal will slide on the floor of the breast without the assistance of sheet-iron, but as the gob will also slide down the pitch and run out with the coal, a different method of working is adopted. (See Fig. 3, Atlas Sheet No. XXII.)

The breast is opened by two shutes driven in from eight to ten yards and the working is then opened out to its full width. Each shute is entirely closed by a plank partition located a few feet from the gangway with an opening in it through which to draw the coal, and which the miners use in passing up into the breast. This partition is called a "battery." When the coal is drawn from the breast through a log battery, the shute battery is known as a "*check battery*." Check batteries are used both to check the flow of coal and to confine the air current to its proper course through the breasts. As the breast is driven up the pitch, the gob is thrown in the center and the coal is thrown down the shute to be loaded in mine cars standing on the gangway below. A manway used also as a shute for the coal, is carried up on each side of the breast to confine the gob, and prevent it from running out with the coal.

In thin workings these shutes may be built of upright props (at right angles to the dip) or they may be inclined "juggler" fashion, but in thick seams the juggler* manway is nearly always adopted. They are faced inside by two-inch plank and are tight enough to make fairly good airways.

This plan of working is not applicable to seams dipping more than thirty-eight or forty degrees, as the miners then have no means of keeping up to the face, (*i. e.*, the slope is too steep to stand on without some artificial means of support,) unless the bed contains such a large amount of refuse that the top of the gob is always within a few feet of the working face.

In its passage down the manway shutes the coal is subjected to much attrition and the waste from fine coal is largely increased. This waste might be greatly reduced if the manways could be kept full of coal and from time to time drawn out at the bottom, as the whole mass would then settle slowly down with comparatively little breakage; but as these passages are needed not only for use as manways but also as airways, this can only be accomplished by driving a manway up between each two breasts to take the place of the juggler manways, and the amount of coal so saved does not warrant the expense of this additional narrow work.

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* The mode of building a "juggler" manway will be described further on.

When the bed is thin, the roof poor, and the dip less than forty degrees, breasts are sometimes opened by a single shute eight or ten yards long, above which the working is opened out to the full width. The coal-way is made up the center of the breast and is kept open by two rows of props on each side, the rows being two or three feet apart and the props from four to eight feet apart. A brattice placed on one of these rows converts one side of the breast into an airway and the space between the other row is used as a travelling-way. The props are of six or eight inch round timber. When this plan is used the roof is so thoroughly propped that accidents from roof-falls are rare. The coal is run as mined, and the gob is piled against the rib on each side of the breast.

When the amount of refuse (gob) is large, or when the bed is gaseous, the breasts are opened by two shutes,—see Fig. 2, Atlas Sheet No. XXII,—which are extended up along the rib to within a few feet of the working face, either by building a juggler manway or by planking carried on upright props. The illustration shows a breast opened by this plan on a dip too steep to enable the miners to keep up to the face; and the platform built on props, on which they stand to mine the coal. On dips of less than thirty-five or forty degrees these platforms are not necessary, and in beds exceeding ten or twelve feet in thickness they cannot be built; hence this method of working is applicable—1. To beds dipping less than thirty-five or forty degrees; and 2. To thin beds on steep dips.

The coal is separated from the refuse on the platform—the coal sent down the manway shutes, and the gob thrown in the center of the breast behind the platform. Breasts worked by this method are spoken of as being worked "*on batteries.*" A considerable amount of gob is purposely thrown on top of the jugglers and manway planking, and a certain quantity of coal is kept on the platform to deaden the blow from falling coal, or roof-falls of slate, that might otherwise crush in the jugglers or props and close the airway, or by breaking the platform would set free a large quantity of coal that would fall and become mixed with the gob below.

The coal is run down the shutes and is loaded into the car over an apron projecting into the gangway.

In the illustration the shutes are timbered, but it will of course be understood that timbering is not erected unless the character of the coal or roof demands it.

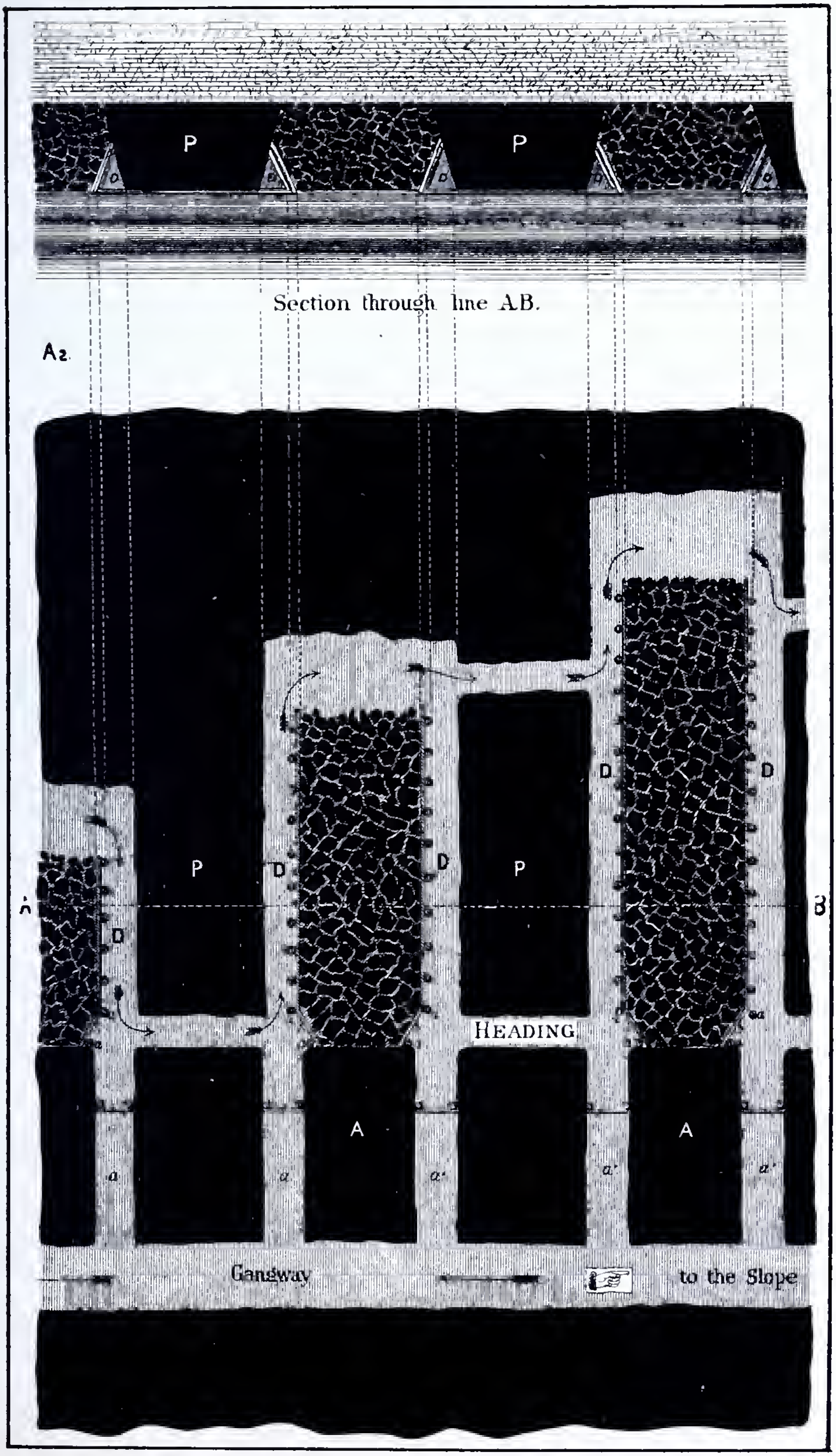
When the dip is steep, forty-five or fifty degrees or more, the coal falls down these shutes with great violence, tumbling around from side to side and in collision with the roof and floor alternately, and a very large percentage of fine coal (*coal-dirt*, *dirt*) is thus made. In the softer, and also in some of the more brittle coals, this is an objection almost fatal to this method of working.

When the amount of refuse is sufficient to fill the center of the breast, the miners can work without the platforms as they can then stand on the gob,—under such circumstances this method can be used in *thick* beds on steep angles of dip.

In addition to these methods of opening breasts on moderate dips, there are several other plans differing from them in minor details, and which will not be separately described here. In some localities the methods especially applicable to thick beds and steep dips have been modified and adapted to thin bed mining and to more gentle pitches, but these adaptations do not reqnire any special discussion.

In thick bed workings on steep pitches, it is impossible for the miner to keep up to the working face,—in other words, he has nothing to stand on,—and it is therefore necessary either to leave the loose coal in the breast or to erect some artificial support. When the bed is comparatively thin the method of working "on batteries" can be adopted, but if the pitch is very steep such a large amount of dirt (fine coal) is made by the coal falling down the manway shutes, that the adoption of this method is not advisable.

As already indicated, the amount of gob may in some


P
P
D
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Section through line AB.
A2.
P
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HEADING
A
A
a
a
a
a
a
Gangway
to the Slope

cases be sufficient to fill the breast and keep the miner up to the face, but the beds now worked seldom contain enough refuse to fill the breasts on steep dips.

When the coal is to be left in the breast to support the miner an entirely different system of opening and working is adopted. It then becomes necessary to provide means for the withdrawal of the coal left in the breast, for the removal of the surplus coal, (the loose coal occupying fifty to ninety per cent. more space than coal in the solid), for ventilation, for the conveyance of supplies and for ingress and egress of the miner.

The surplus coal may be drawn out at the bottom through the opening in the battery, but it is frequently sent down the manways, which are also used for ventilation and as traveling-ways. These manways are often called manway-shutes or juggler manways, this latter name being given to them because they are built of "*jugglers*" or inclined props, faced by two inch plank. They are made as nearly air-tight as possible, especially the one conveying air to the face.

A section at the top of Page plate No. 18 shows how these manways are carried up along each side of the breast.

The jugglers are notched into the pillar coal and also into the floor to hold them securely in place, and are set at intervals of from four to six feet.

The surplus coal may, as already indicated, be drawn out through the battery hole; but, if the roof of the bed is poor, the movement of coal through the breast may cause it to fall and mix with the coal; if the floor is soft, the jugglers (being stepped down into the floor) may be unseated and the manways closed or the ventilation impaired. For these reasons the surplus coal is best sent down the manways and the loose coal in the breast allowed to lie entirely undisturbed until the breast is driven to the limit.

When the surplus coal is to be drawn out through the battery, the plan of opening by one (central) shute is preferred, because the movement then takes place principally in the coal lying near the center of the breast, and the risk of unseating the jugglers is much less than when the breast is opened by two shutes.

To prevent the coal from running out through the shute, the opening into the breast is closed by a "battery" constructed by laying three, four, or five heavy logs across the opening as shown by Fig. 1 Atlas Sheet No. XXIII, or built on props as in Fig. 3, Atlas Sheet No. XXI; a hole is left in the center or at one side of the battery through which the coal may be drawn. The battery closes all of the opening into the breast except the space occupied by the juggler manway, and is made as nearly air-tight as possible by a covering of plank.

Double shute breasts.

When the Mammoth bed is very thick it is frequently worked on steep pitches by opening the breasts by two main coal shutes, each of which is provided with a battery through which the coal is drawn; a manway shute is driven up through the center of the pillar for a few yards and is then branched out in both directions until the branches,—called "Slant shutes,"—intersect the foot of each breast near the battery as shown by Fig. 3, Atlas Sheet No. XXI. The juggler manways are started at this point and continued up on each side of the breast. The main return airway is driven in the solid, through the stumps above the gangway.

In this illustration the gangway is shown driven against the roof,—a plan not generally adopted. When, however, the pitch is very steep the coal in the loading shute is more readily controlled, if the pitch of the shute is lessened by driving the gangway along the top. When this is not done, a gate placed in the shute below the check battery enables the loader to properly handle the coal. Coal in excess of the amount necessary to keep the miner up to the face may either be drawn from the main batteries, or sent down the manways and slant shutes to the manway shute, from which it is loaded through an air-tight check battery.

To give the loader sufficient working room the main shutes are usually eight or nine feet wide, but sometimes only for the first six or eight feet; above this they are driven about six feet wide by five or six feet high. The manway and slant shutes are driven six feet wide by four to six feet high (measured at right angles to their inclination).

In this plan the miners always have free access to the breast, the traveling-way being entirely independent of the main loading shutes; and the loader is not troubled by the manway coal, as this goes down into the slant shutes, so that he can work at the battery when coal is coming down the manway.

When it is desirable to run the stock coal from the breast as quickly as possible to empty the breast and avoid roof-falls or slides from the floor, the coal can be drawn from both shutes at once. This is an advantage possessed in common by all double shute breasts over those opened by a single shute.

In this and other illustrations reproduced from the Report A^2, A is the stump, and P the pillar left between the breasts.

As the number of openings—three for each breast—greatly weakens the gangway stumps and pillars, this plan is best adapted to workings in strong coal.

When the bed is not thick enough to carry an independent airway over the gangway, the shutes are driven up as in the plan just described, for a distance of about ten yards, or until they intersect the heading (airway). The breast is opened out just above the heading, a battery being built in the heading immediately above each shute. A manway is started from the gangway and driven up through the center of the pillar stump until it intersects the heading, and a trap-door is placed at this point to confine the air, and to prevent accidents from objects falling from the heading into the gangway below. This manway is made sufficiently large to be used as a traveling-way and as a passage for timber and supplies,—generally about four by six feet, or perhaps somewhat smaller.

In this, as also in the preceding plan, it is not necessary to drive the main shutes more than six feet wide, but when this width is adopted the lower portion of the shute may be broadened out to about nine feet to give the loader sufficient working room.

In the plan last described the manway coal comes down into the main shutes, and often occasions great annoyance to the loader. The plan is also open to the same objection with the preceding plan of necessitating three openings for each breast.

A further modification of the same plan is effected by replacing the manway driven through the pillar stump by a manway planked off on one side of one of the main shutes. The shute carrying this manway is driven three feet wider than its mate. The manway is protected from coal coming down the breast manway by an inclined plank covering which directs the manway coal into the main shute.

This is not a good plan for use in gaseous workings as it is almost impossible to prevent a very large amount of leakage through manways so constructed

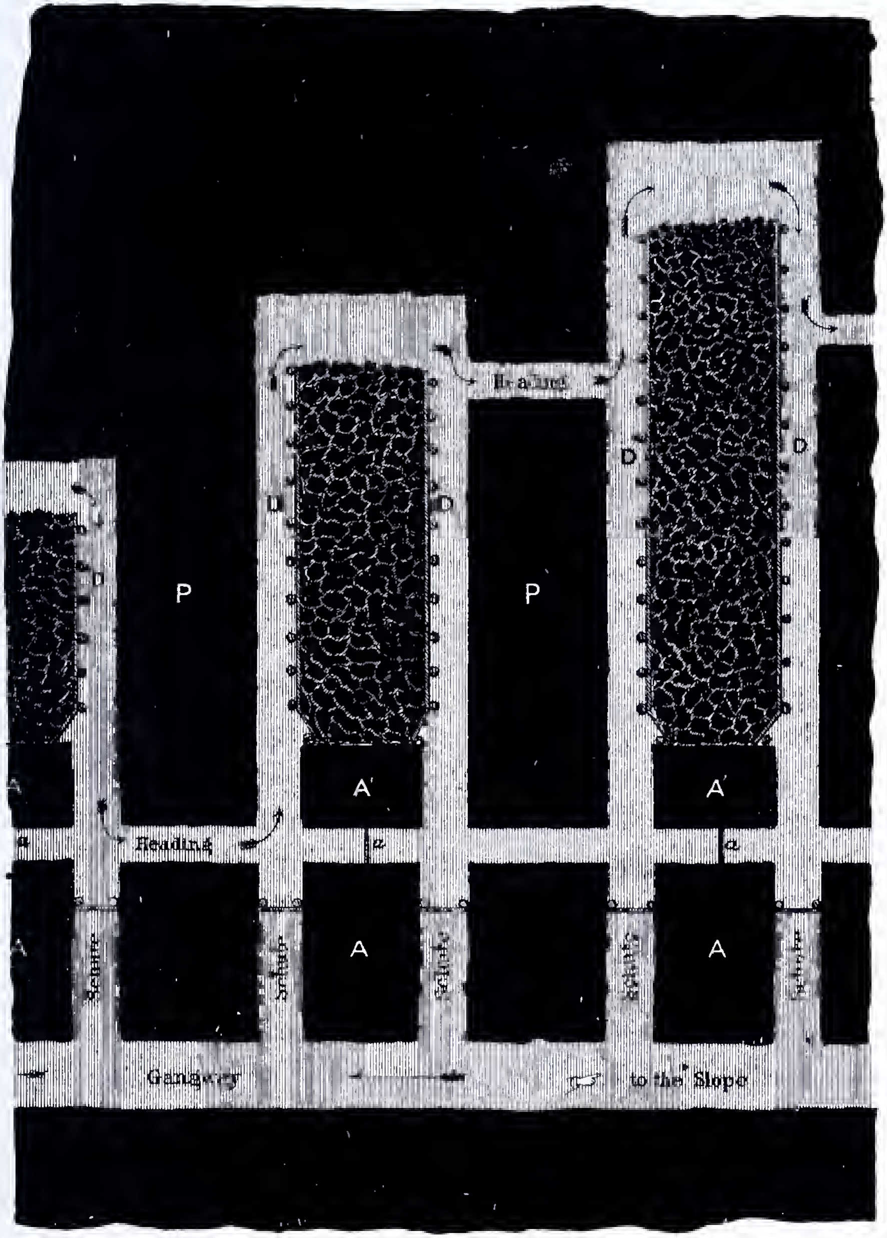
A plan less complicated than any of the above-described methods is shown by Page Plate No. 18.

In this plan the main shutes are driven up to the heading from which the breast is opened out, a log battery being built at the top of each shute at the points marked a a, a a, in the illustration. The shutes are used not only for drawing the battery coal, but they also receive the manway coal, and are used as travelling-ways and for the conveyance of supplies.

In this, as in all of the preceding methods a check battery is placed in the shute to prevent the air current from taking a short cut from the gangway through the shute to the breast airways, and this check battery is of great assistance to the loader when the shute has a very steep pitch, as he can readily control the flow of coal through the drawhole. The drawhole is sometimes closed by a piece of brattice cloth, hung so as to form a flap, under which the coal freely runs.

All of these methods are open to the objection that in case of any accident to the breast manways by which the flow of air is obstructed, there is no means of isolating the breast in which the accident occurs, and the ventilation of all the breasts beyond it is impaired or entirely stopped.

Double Shute Breasts with Stump.



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A2.

The consequences of such an accident in gaseous workings may be most disastrous.

To overcome this difficulty the plan shown by Page Plate No. 19 is sometimes adopted. In this method of opening the breasts are not opened out from the heading, but the heading (airway) is kept intact by leaving a small stump (A′) above it. If any accident occurs to the manways D, D, the breast may be isolated and the air current taken directly to the next breast by removing the bratticed stopping at *a*, *a*. As the coal left in the stump A′ can be mined after the breast is exhausted and all the stock coal has been drawn from it, or even after the pillars have been skipped, the amount of coal recovered in this plan of working is fully as great if not greater than by the other methods.

To insure the safety of the gangway these breast stumps may be left in until the pillars are robbed out and they are reached in the process of robbing out the gangway.

Single shute-breasts.

When the coal gives off large quantities of gas it is desirable to perfect the ventilation by making the breasts as nearly air-tight as possible, and this object can sometimes be more thoroughly accomplished if the breast is opened by a single shute.

When the bed is thick enough to carry an airway over the gangway the plan shown by Fig. 4, Atlas Sheet No. XXI, is frequently adopted. The breasts are opened by a shute nine by six feet, driven up the pitch,—or if the gangway is driven along the top, across the bed,—a distance depending on the dip, but usually from eight to twelve yards, and the breast is then gradually widened out as shown by the illustration. In the center of each pillar a small manway shute is started and after being driven up a few yards is branched out in both directions until it intersects each breast. From the top of these branches manways are carried up on each side of the breast as in other plans of opening. In this respect this plan resembles that shown by Fig. 3, Atlas Sheet No. XXI, but it will be observed that the main shute and battery have no connection with the slant and manway shutes.

A narrow travelling-way (manway) is usually made by planking off a portion of the main shute so that the starter (loader) can have free access to the battery at all times.

It will be observed that in this plan of working the manway coal can be sent down through the slant shutes and loaded through an air-tight check battery placed in the manway shute near the gangway, or the surplus coal may be drawn out from time to time through the main battery; and this can always be done in breasts opened by a single central shute with much less risk of displacing the manway props, than in double shute breasts in which the movement of the loose coal is principally along the *sides* of the breast instead of through the center.

When the dip of the coal is much more than fifty degrees, the gangway is not infrequently driven in the top bench, as the risk of closure from a squeeze is much lessened, and the shutes may be driven at an inclination on which the coal is more easily handled; but this plan is objectionable because it increases the cost of opening a breast by greatly lengthening the main shutes and the manway and slant shutes, as shown by the illustration, (see Fig. 4, Atlas Sheet No. XXI).

A small airway marked F is driven from the airway C to the manway shute at F; but cross-headings between the airway and gangway are also necessary when these headings are long and make a large amount of gas while being driven.

The small airway F and the airway C are not used when the breasts are working; but if any accident happens to a breast manway by which the ventilation is impaired, the air can be conveyed around the breast through the airways F and C, by simply removing the stoppings.

This plan is especially adapted to working thick, steep pitching beds of soft, gaseous coal. Its advantages may be summarized thus:

1. Workings easily made nearly air-tight;

2. Stock coal is drawn from the center of the breast;
3. Manway coal sent out through slant shutes;
4. Main battery entirely independent of manways and slant shutes;
5. The air current can be conveyed *around* the breast in case of accident;
6. Only *two openings* through the pillar and stump coal are made in opening each breast;
7. A permanent return air-course for the air after the breasts are exhausted and closed.

Its disadvantages are:

1. The slant shutes and main shutes are sometimes very long;
2. The stock coal cannot be drawn as quickly as from breasts opened by two main shutes;
3. The total amount of dead-work is often very much greater than in opening by other methods.

The plan shown by Page-plate No. 20 is somewhat similar to the double shute plan of Page-plate No. 18, but it has one great advantage over this latter method, viz., the possibility of isolating any breast in case of accident, or during alterations or repairs. In this plan the breast is opened by a single central shute driven up to the heading, and the breast is gradually widened out above the heading, as shown by the illustration. The battery is built on three props set across the opening in the upper side of the heading, and is provided with one or two draw-holes. A stopping, "a," is placed across the heading in line with the center of the battery.

As the manway coal is sent down into the main shute, a separate travelling-way must be provided, and this is made by driving a manway up through the center of each gangway stump to the heading. This passage is provided with a trap door to hold the air.

It will be observed that this plan accomplishes the same object as the plan shown by Page-plate No. 19, viz., the heading is kept open for use as an aircourse, so that in case of accident to any breast manway, that breast can be isolated by removing the stopping *a*.

A still more simple plan of opening single shute breasts is shown by Fig. 1, Atlas Sheet No. XXIII, which is largely used in the shallow workings of the small Lehigh basins. The shute is driven up in the center about nine feet by six, for a distance of eight or ten yards, and the breast is then gradually widened out until its full width is reached, when a cross-heading is driven through to the adjoining breast to establish ventilation.

A plank manway is carried up on one side of the shute to the battery, which is placed at the top, and the breast manways are started from each side of the battery. It is evident that breasts opened on this plan cannot be made tight, and the leakage of air is usually too large to permit its use in gaseous workings.

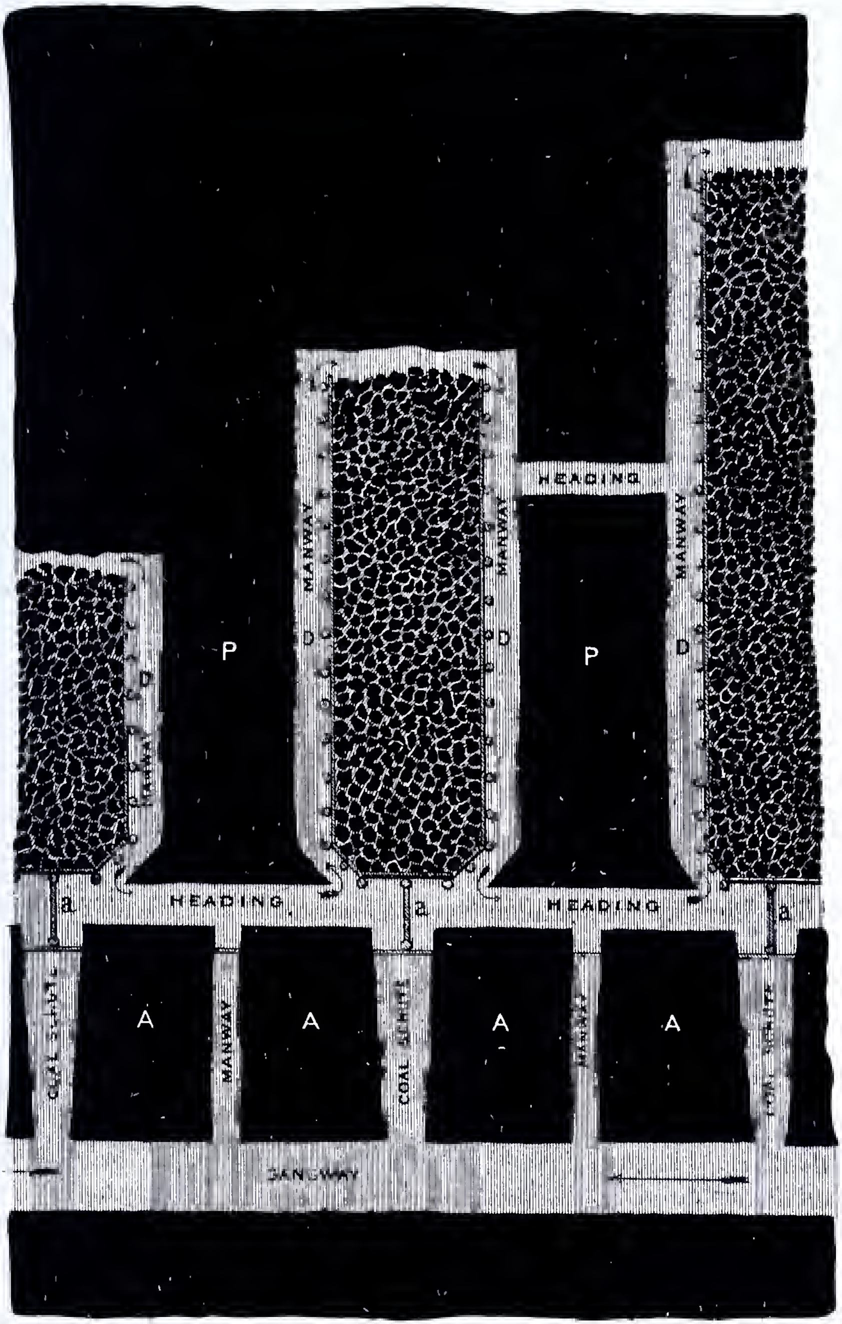
As the weight of the stock coal must be largely supported by the battery and manway props, the plan is not a good one for steep pitching breasts. It is also open to the objection that the breasts cannot be isolated in case of accident; but as it is the cheapest method by which a thick, steep pitching bed can be opened, these disadvantages do not prevent its adoption. At mines making only a small amount of gas, and especially on moderate pitches, it is a very satisfactory method of working.

Single shute breasts are not always opened by a central shute, but the main draw-shute is sometimes located on one side of the central line, as shown by Fig. 52, or even directly in line with one side of the breast. When this plan is adopted a manway shute is usually driven up in line with the other side of the breast to take the manway coal and furnish ready access to the workings, as shown by Fig. 53.

These illustrations show a method which has been largely used by the Lehigh Coal and Navigation company in the Panther Creek basin.

The main shute is driven up to the heading, and a travelling-way planked off on one side; the breast is opened from the heading and the battery is built a little to one side of the top of the shute, and a triangular stump is left above to protect the heading and to direct the flow of coal

Breasts with Single Shute.



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towards the draw-hole. Figs. 53 and 54 show a method lately adopted by which each breast is given a separate split of air, and is rendered entirely independent of the adjacent workings. In Fig. 54, *e* is the gangway, *s* the shute, *c* the return airway, *m* the cross-cut airway, and *a* the heading. It will be observed that the gangway doors are dispensed with in this plan, and there are no cross-headings connecting the breasts.

In Fig. 53 the airway is drawn on the wrong side of the gangway,* and the cross-cut airways appear much longer than they really are; their true length is shown by *m*, Fig. 54.

In workings of this class the surplus coal is not infrequently drawn out through the battery; when opened without the manway shutes, and the surplus coal is not drawn from the battery, it must all be sent down the manway on stump side of the breast.

A somewhat different arrangement is effected by starting the manway shute from the lower end of the main shute of the adjoining breast, thus lessening its inclination and making it a more convenient travelling-way, and greatly decreasing the labor of raising the timber and other supplies. Such an arrangement is of course only suitable for very steep dips, and on such it has been found advantageous to slant or curve the lower part of the main shute to check the descent of coal from the battery. When the main shute is empty the coal falls on dips of 60° and upwards with nearly as much violence as in a shaft, and a curve or slant near the bottom of the shute not only protects the check battery from damage, but also prevents the coal from jamming fast and blocking up the shute. On these steep dips it is not advisable to build a manway in the main shute as it is costly to build and difficult to keep air-tight; the manway shute is therefore used as a travelling-way to the adjacent manways of two contiguous breasts.

When the floor or coal is very soft, and the jugglers are easily unseated, necessitating continual repairs to keep the manways open, they may be replaced by a manway driven up through each pillar and connected with the breasts by driving headings to the right and left alternately. The manway may be driven up from either the gangway or airway(?), and may be located in either the top or the middle bench of the bed; but when not driven near the top, the headings are driven on a slant, so that when they intersect the breast they are in the top bench. When so driven they have sufficient inclination to be used as shutes, and the surplus coal can be sent through them to the manway and loaded from an air-tight check battery placed near the gangway.

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*This was purposely done to simplify the diagram.

The object of opening the headings into the breasts in the top bench is to prevent them from being too quickly closed (covered up) by the loose coal in the breast.

Almost any of the above-described plans may be modified in this way, but the cost of opening is much increased by the driving of the pillar manway and headings.

In addition to these methods of opening and working steep-pitching breasts several other plans have been devised, some of which have been tried and found worthless, and others no better than those in common use, while a few still continue to be used in a small way; but as these latter differ only in their details or are simply modifications of those already described, they need not be considered here.

It will be observed that in all of these methods of working, the slate, bone, "sulphur," and any other refuse must be taken out of the mine with the coal, while in workings opened on more gentle pitches only the coal is sent out, the refuse being left in the breast.

As the coal cannot be cleaned in the breast, and as a large amount of stock coal is left in the breast to be drawn when the working is exhausted, a different system of payment is adopted in breasts worked "*By-the-run.*"

The miner receives a certain allowance for opening the breast, driving the headings, shutes, etc., which varies with the amount of dead-work and the character and thickness of the coal; and is then paid, not by the wagon, but a certain sum *per lineal yard of breast driven*,—this method of payment is called working "*By-the-run*," in contradistinction to the method of payment "*By-the-wagon*" adopted in workings flat enough to enable the miner to keep up to the face without keeping the breast full of loose coal.

At some collieries, however, the miner is paid by the car for coal worked "by the run."

The *cost of opening* pitching breasts depends largely upon the amount of timbering necessary; this is governed by the character of the coal or roof. In some workings almost no timber is used except for juggler manways; in other localities all of the shutes, manways and even cross-headings must be securely timbered. As the cost of timbering depends not so much on the plan of opening as upon the character of the coal, we may disregard it in considering the relative cost of opening by the plans above described.

The method shown by Fig. 1, Atlas Sheet No. XXIII, is the cheapest plan.

The other single shute plans follow next in order and the double shute plans are the most expensive methods of opening; that shown by Fig. 3, Atlas Plate XXI, and the plan shown by Page plate No. 19, when opened with a manway through the pillar stump, cost more than any plans in common use.

Shutes six feet by six can be driven for an average cost of two and half to three and a half dollars per yard; six feet by nine, at a cost of three and a half dollars or more per yard.

Manways cost from two to three dollars per yard to drive, and cross-headings about the same amount.

A plan rarely used, but which may be advantageously adopted when the roof is very bad and the coal lies on a moderate dip, is shown by Fig. 5, Atlas Sheet No. XXII. As I have not seen any breasts worked on this plan I quote Mr. J. P. Wetherill's description* of it.

"A shute is driven from the gangway up the pitch as far as it is desired to work the coal, and timbered just as the gangway is timbered except that it is not so large. As indicated in the drawing, mining is begun at the extreme end of this shute, the breast being opened out to full width and worked back towards the gangway.

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* Report A^2, page 16.

"The advantage secured is that between the miner and gangway there is always the timbered shute as an outlet for himself and the coal he mines, which could not be maintained if the full width of the breast were driven as in the preceding cases, owing to the bad top.

"Props of six to eight-inch timber are set close to the face of the coal which is being mined, and are sufficiently strong to prevent the top breaking so close to the face as to injure the miner, while in the exhausted space it is allowed to fall at pleasure.

"If the pitch were so steep that this fallen top would slide on the bottom down to the face of coal on which the men were at work, this plan could not be used, and the pitch selected in the illustration is such as to require sheet iron in the shutes, for the coal to pass over to the gangway."

It will be observed that this plan stands midway between the *pillar-and-breast* system and *long-wall withdrawing*. The cost of opening is greatly increased, not only by the cost of the large shute, but by the increased length (nearly double) of all the cross-headings.

Reworking old breasts.

At many old collieries on the Mammoth bed, only the lowest or best bench of the bed was worked, and the breasts were not worked up as close to the overlying gangway as in the present practice.

Many of these old workings have been re-opened and a large amount of coal reclaimed that only a few years ago was supposed to be entirely ruined.

When the old gangway cannot be re-opened and retimbered, a new gangway may be driven as close to the old one as possible, but it is nearly always cheaper to re-open the old passage. If the breasts are not entirely closed by roof-falls the plan shown by Fig. 4, Atlas Sheet No. XXIII, is adopted.

A thin skip is taken off the rib on each side, or one thick skip from the pillar on one side of the breast, and if the chain pillar between the workings and the overlying gangway is thick, the breasts are driven up a certain distance and then narrowed to a point as shown by the illustration, or they may be holed through into the gangway, unless it is desirable to preserve this as a water course.

It is of course necessary to mine the coal from any upper or lower benches that may have been left untouched by the old workings, and to extend the breasts up to the limit, before skipping the pillars.

This is the plan by which the coal left in the old Sugar Loaf No. 2 workings at Hazleton is to be recovered.

The cost of reworking by this method will often be greatly increased by the necessity of removing a large amount of gob from the old breasts, and this is especially the case at mines in which the lower benches have been left untouched by the old workings.

Even when old workings of this class are completely closed by roof-falls, they can doubtless be profitably reworked in the future.

As at many of these old collieries the pillars are thick we may expect to be able to recover a large amount of the imprisoned coal by driving new gangways and opening breasts in the pillar coal, and when this plan cannot be adopted, resort will doubtless be had to the Rock-shute method.

In re-opening old workings of this class, trouble from fire-damp is seldom experienced, because these workings are all comparatively shallow and the quantity of gas originally given off by the coal must have been quite small.

Black-damp (carbonic acid gas) is, however, found in large quantities, and often seriously retards the work of re-opening.

The occurrence of standing water in old workings may also be a cause of more or less difficulty, especially when no reliable map of the workings exists. Such water is nearly always exceedingly rank with acid, and attacks the column pipe or any other iron with which it comes in contact, with incredible rapidity.

Probably the greatest difficulty in the way of opening and reworking many of these old workings is the absence of reliable maps. It frequently happens that no maps exist; in some cases none were ever made, or they have been lost, or destroyed by fire. When maps are in existence they frequently do not show the workings as they exist at present, either because they were not kept up to date, or because they do not show the areas that were robbed before the workings were abandoned.

In many cases the only information that can be obtained is from the memory of old decrepit miners who were employed in the workings in their boyhood.

CHAPTER X.

Coal Mining Tools and Methods.

Anthracite mining requires the employment of a method differing widely from the system ordinarily used in the extraction of bituminous coal.

The operation of "*cutting*," "*getting*," "*winning*," "*breaking*," or "*mining*" bituminous coal is commonly effected by making an undercut with a coal pick in the bottom bench, the floor, or a parting slate near the bottom of the bed, and then breaking down the top benches by wedging, by blasting, or with a pick.

The undercut is from two and a half to five feet deep, and when made with a miners' pick is from six to twelve inches high at the face, and about one inch wide (the breadth of the cutting point of the pick) at the other end. In making the cut the miner usually lies in a half recumbent attitude, or flat on his side, with his head close to the floor to enable him to see into the cut.

Anthracite coal is not only too hard to be readily undercut in this way, but it is so brittle, and the splinters flying off from the coal are so sharp, that few miners can be found who would be willing to expose their hands, face, and eyes to the flying fragments.

Machines for undercutting the coal have replaced the ordinary method of underholing by hand at a considerable number of bituminous mines in this country. These machines consist of some form of engine,—rotary or oscillating, but more generally of the horizontal type,—and a cutting device.

The cutter is either a large horizontal wheel with cutting teeth on its periphery, an endless chain carrying a series of teeth, or a revolving bar about two inches in diameter and three or four feet long, with teeth arranged alternately in three or four rows throughout its length. Attempts have also been made to introduce machines in which the cutter acts like a saw. A very different principle is employed in the Harrison machine, which is virtually a percussion drill arranged for use as a horizontal channeling machine.

All coal cutting machines that have been successfully used are driven by compressed air or steam; those in which it was proposed to drive by hand-power have not given good results.

The claim made that some of these machines will make an undercut three feet deep and from twenty to fifty yards long in one day of ten hours, seems to have been well sustained by practical tests, of some, at least, of the various forms of coal cutting machines.

But the mining of anthracite presents difficulties which have not yet been overcome by any of these machines. The coal is too hard to be readily *scraped off* by rotating cutters, and the percussion cutters are not, in their present form, adapted to the work.

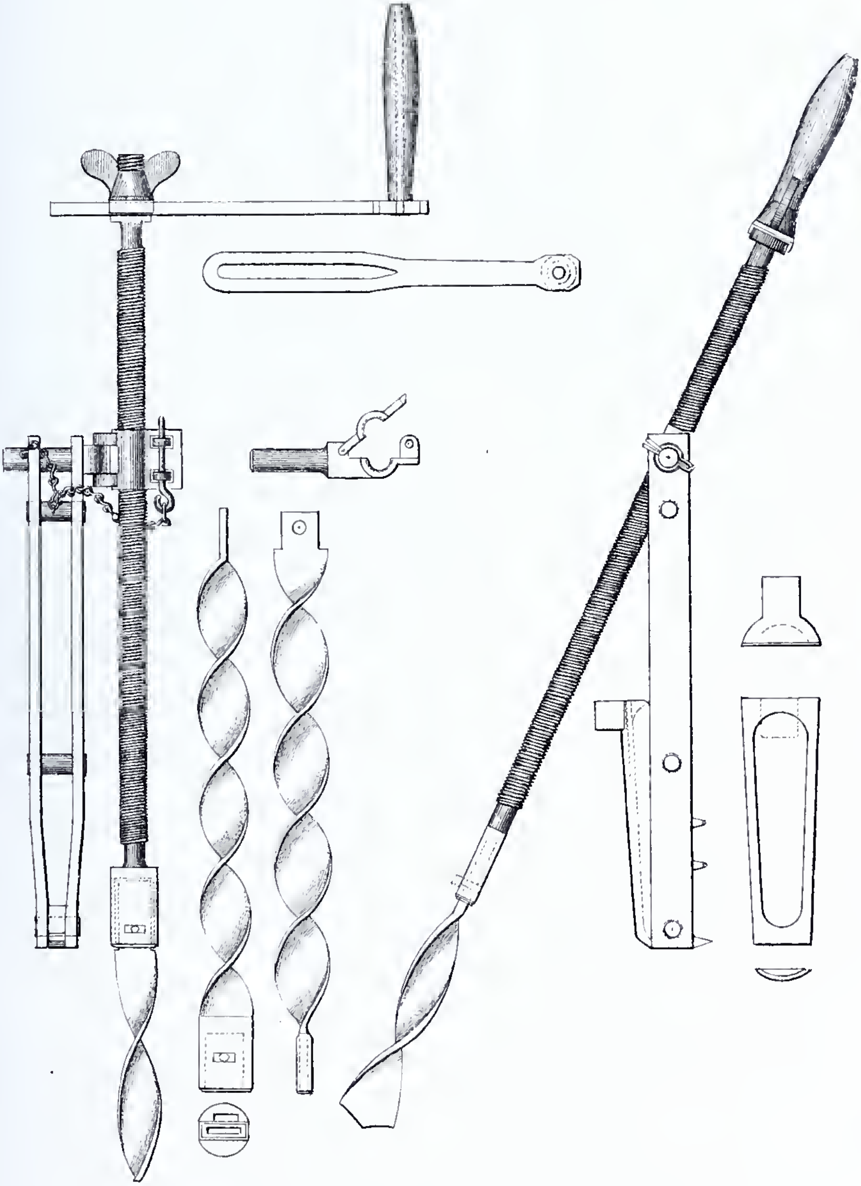
In addition to the difficulty of making machines to cut coal as hard as anthracite with sufficient rapidity—which could doubtless be overcome—the steep pitches on which a large percentage of the coal must be mined, make it very difficult to adjust the machine and hold it up to its work.

These are some of the reasons why coal cutting machines are not used in anthracite mining. Several manufacturers of these machines have been offered all facilities for making practical trials, with promises that if experiments were satisfactory, a certain number of the machines would be taken on trial; but after visiting the mines and seeing (for the first time) the peculiar difficulties of anthracite mining, these gentlemen did not send their machines for trial.

HAND DRILL

designed by

F. B. PARRISH, M.E.



Scale

Under-holing is employed in mining thick anthracite beds, but instead of making a thin cut near the bottom with a pick or machine, from three to six or seven feet of coal is removed by shots located in the bottom bench, and the under-holing is often continued in a considerable distance before blowing down the top benches. In some workings the breast is opened out and driven to the limit in the lower bench of coal, and the top is then attacked and worked back.

But the bottom coal is not always the part first attacked ; sometimes the top coal is taken out to a depth of a few feet and the bottom benches blown up afterward.

In bituminous mining the direction of the *cleat* frequently determines the direction in which the working faces must be advanced in order to mine a maximum amount of coal with a minimum amount of labor, and to reduce the amount of small coal to a minimum ; but in the methods of working breasts adopted in anthracite mining, the steep dip of the measures determines the course in which the breasts *must* be driven, and it is therefore impossible to lay out the workings so that the working faces shall be driven in a direction predetermined from the direction of the cleat.

But it will be observed that while the method of working bears no relation to the cleat*, it is often possible for the miner to take advantage of the cleavage joints* in locating shots so that the number of holes is reduced, and the efficiency of each shot increased. This is often done by keeping one side of the face in advance of the other, so that the working face runs diagonally across the cleat,* or cleavage.

In mining, as in tunnel driving, or any other species of work requiring blasting, the location of the hole is always more or less governed by the existence of natural fissures, slips, or cleavage planes, and by cracks or fissures made by the last set of shots.

Several different forms of hand borers are now in use, but the ordinary drill is still the main reliance of the majority of miners. The coal drill shown by Figs. 41 and 42 is simply a round or hexagonal bar with a cutting edge dressed very like an ordinary rock drill, but the edge is generally spread more and tapers very gradually,—in other words it is a sharp, not a blunt, chisel cutting edge.

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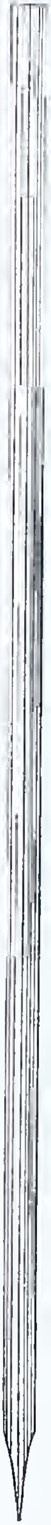
* Anthracite coal exhibits to a very small degree the cleavage planes, which in bituminous coal constitutes the "*cleat*"; but the main cleavage joints are often much more pronounced than in bituminous coal. See Chapter I.

The drill is used as a *jumper*, that is, it is "jumped" in and out of the hole by the miner instead of being struck on the head with sledges, and to give the most effective blow the miner is forced to adopt a position suited to the location and direction of the hole. As the hole may be directly overhead and at any angle, or immediately under foot, or in a vertical face and at any elevation, knee high, breast high, or above the miner's head, and at any angle of inclination, there is no one posture best adapted to this work; the miner simply places himself in the position in which he can most readily and effectually handle the drill.

The various forms of hand-boring coal drills which are now successfully used in anthracite mining, although covered by different patents, are all nothing more nor less than the old *slate auger* or *shale auger*, which has been known and used for many years in mining soft shale and slate.

These borers or drills vary from each other only in the details of their construction, and it is these that the patent rights cover. The method of lengthening the drill, of feeding it forwards, of adjusting the crank, or of gearing, of fastening the drill in the coal or otherwise to hold it up to its work,—these are the principle details in which the ma-



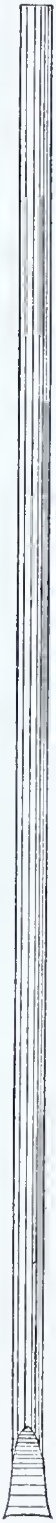


Fig. 40.

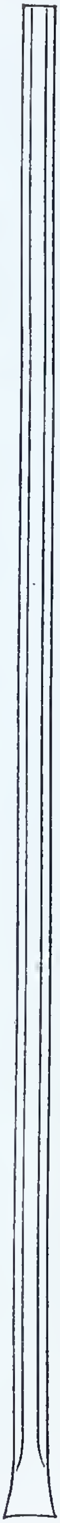
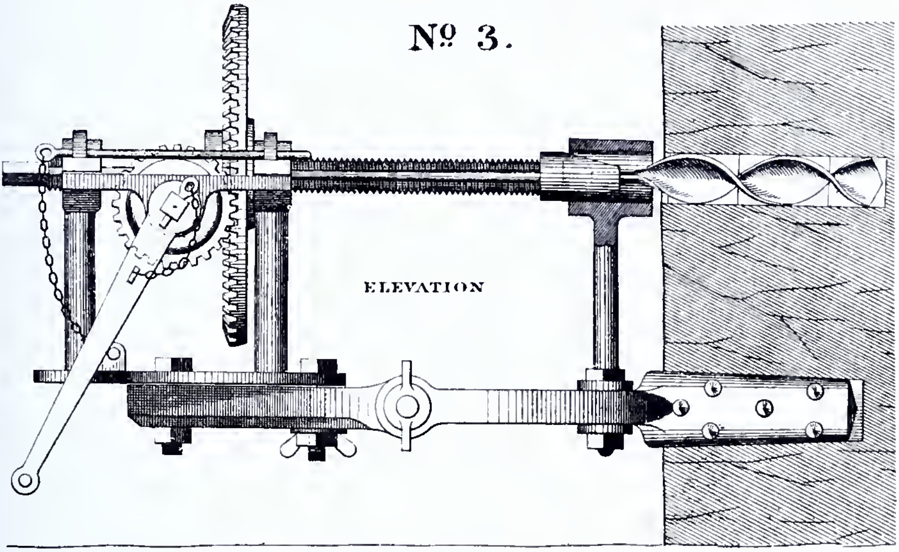


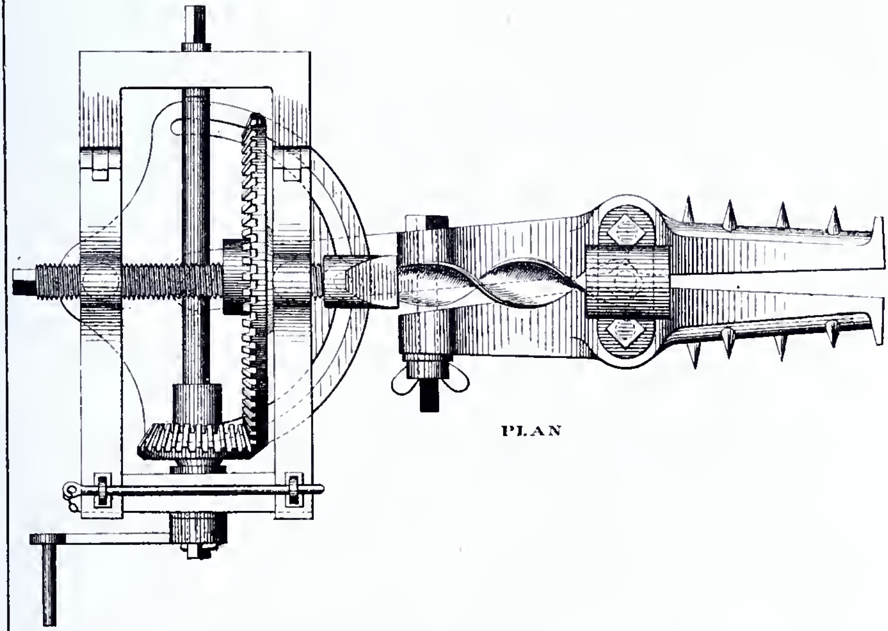
Fig. 41

THE HOWELLS MINING DRILL.

Nº 3.



ELEVATION



PLAN

chines vary from each other. There are also differences in the pitch of auger, in the shape of the cutting edges or bit, etc., etc., but the principle employed is the same in all,—they are simply *augers*.

Probably the most widely known of all these drills are the drills made by the Howell's Manufacturing company, and known as the Howell's drill.

Four styles of the Howell's drill are made, three of which, Nos. 0, 1, and 2, are very similar machines, differing principally in size and strength, but the No. 3 drill is a very different machine.

Page plate No. 22 shows the method of gearing the power up in the No. 3 drill. The drawing also shows the clamping device, which is the same on all of these machines.

This machine is intended for driving through slate or fireclay, shale, bony coal, or for gangway and airway driving. The plate shows the machine as made with a much smaller clamp arm than is used at present.

At the bottom of Page plate 23 the Howells' drill No. 2 is shown by a small cut. The principal features of this machine are similar to the No. 3 drill, except in the gearing, the crank being attached directly to the main screw shaft. The Nos. 0 and 1 drills are of similar construction, but are smaller than this machine.

The Providence drill shown on Page plate No. 23 is a less complicated machine. This drill is clamped by wedging. It will be observed that in this machine the auger is supported at only one point.

In the McMurtrie drill the auger works through the *center of the clamp*, which is hollow. It therefore follows that only one hole can be bored by the drill each time it is set and clamped, while with some of the other drills one clamping hole may be used to hold the drill while two or three holes are being bored.

A drill invented by Mr. F. B. Parrish is shown by Page plate No. 21. It is similar in many respects to the Providence and Howell's drills, and as its construction is seen at a glance, a detailed description of it is unnecessary.

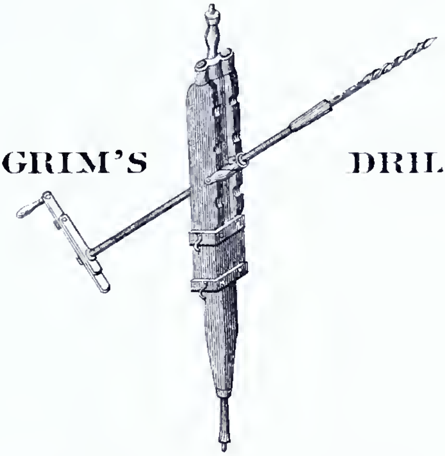
In using all of the drills above described, the miner must first hew or drill a hole in the face three or four inches in diameter, or two to four inches high by five to eight or nine inches wide, and six inches to one foot deep. The clamping bar is firmly fastened by being wedged or clamped in this hole. When the drill is supported at only one point, as in the Providence drill, it is also necessary, or at least advisable, to start the bore hole by drilling (with an ordinary drill or pick), or to make a little pot shaped depression in the coal before commencing to bore. This will not be necessary in using McMurtrie's drill, or any form in which the drill is held in place by a guide.

A very different style of drill is shown at the top of Page plate No. 23, in which the drill is held up to its work by a bar fastened between the roof and floor. The weight of the bar, and the impossiblity of fixing drills of this class in thick seam workings, will probably act as an effecual bar to their adoption in anthracite mining.

The advantages derived from the use of these *boring tools* to do work otherwise performed with ordinary drills, have not as yet effected any marked reduction in the cost of mining. But it does not follow that advantages have not been derived from the use of these machines, nor that the operators will not eventually be benefited by a considerable reduction in the cost of mining, effected by the general adoption of this system of boring. But up to the present time *the miners* who have used these machines have received nearly all the benefits accruing from their use.

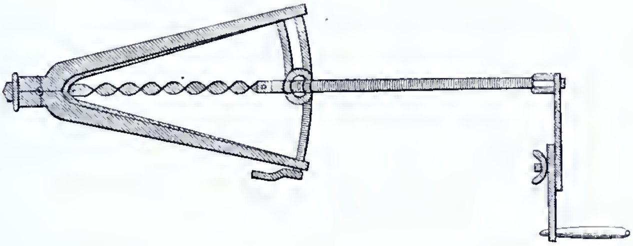
As the inevitable tendency of inventions whereby mechanics or laborers are enabled to do more work, is to reduce the cost of that work; or, in other words, as the miner or mechanic doing work requiring a certain degree of skill will not receive more than a certain sum per day for that work, without regard to the increased amount of work improved machinery may enable him to perform, the benefits must eventually accrue to the operators and consumers.

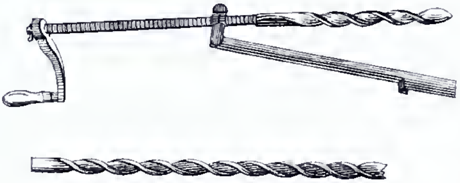
But at the present time the increased daily output from the use of these machines has not been sufficient to warrant a reduction in the price paid for mining coal, and the miner who owns and uses machines by which he can more easily



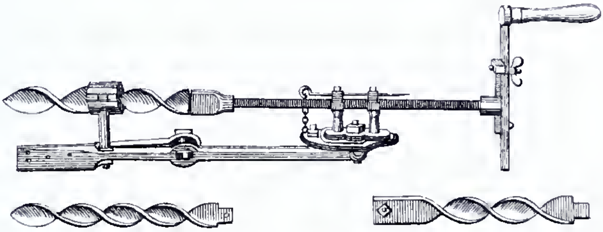
GRIM'S DRILL

Mc MURTRIE DRILL





THE PROVIDENCE DRILL



HOWELLS DRILL Nº 2

mine sufficient coal to make good wages, (receiving the same price for mining as his more conservative comrades who still adhere to the old drilling method,) is benefited either by receiving pay on a larger number of cars, by finishing his work for the day at an earlier hour, or by not being forced to work as hard as those using the ordinary drill.

Some miners cannot learn to use these boring machines; they can neither set them properly, hold them in position until the hole is finished, nor bore as quickly with them as with the ordinary hand drill. Others will not use them because they dislike the labor of turning the crank. It must be admitted that this last objection is well founded, and that operating a "*jumper*" is generally a more pleasant kind of work than turning the crank of a hand drill.

These drills are commonly the private property of the miner, and before he can be induced to purchase one, he must be convinced not only that it will be to his own advantage to own it, but he must be persuaded to abandon the spirit of conservatism common to tradesmen of all classes,—often a very difficult, if not impossible, matter.

To obtain the best results from these machines, or even to make as good time as with the ordinary drill, they must be placed in the hands of apt, intelligent, and experienced miners. In the hands of a dull, slow thinker, less work will be accomplished than with an ordinary drill, but a good workman can often drill his holes with the machine in half the time occupied in drilling by hand. These are some of the causes why boring drills have not more quickly replaced the hand-drills, and as these causes are still operative, it will probably be many years before even a majority of our miners throw aside the old tools for the new.

As there is at present more or less rivalry between the manufacturers of drills of this class, and as injustice might be done to some by any expression of personal opinion as to the best drill now made,* I will merely indicate some of the "*points*" by which the miner is usually influenced in selecting a drill, viz: The cost, weight, bulk, the facility with which it may be set and a hole started, efficiency of the clamping device, wearing quality of the bits, power required to run it, the facility with which it may be used in different positions, and the wearing qualities of the machine as a whole. The *efficiency* of the machine, in other words, the speed of boring, depends principally upon the quality and tempering of the bit and the shape in which the cutting edges are dressed, and not upon any peculiarity in the design of the machine.

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* In addition to those illustrated in this report there are several other forms now manufactured under different patents, but not differing in principle from those above described.

For tough slate and rock work, drills in which the power is geared (1 to 3, to 6) are successful where other drills are useless, but in drills of this class the power applied may often produce too great a tortional strain on the auger, and the advantages of having the power increased by gearing is handicapped by the necessity of limiting the power applied, to the strength of the auger.*

The method of lengthening the auger by inserting a segment between the bit and the feed screw is clearly shown by the illustrations. As these machines are designed to replace the method of drilling by hand, but not to effect any change in the method of locating, changing, or firing the holes, the bits are made to bore a hole of about the same size as by the ordinary method.

Powder.

Black Powder has been shown by long experience to give the best results in anthracite mining, both in its efficiency as compared with other explosives, and especially in producing a minimum quantity of fine coal.

Higher explosives of the dynamite family, which produce no flame, are not infrequently used in gaseous workings, and also in driving gangways and air-courses when the coal is very hard or gaseous.

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* I have used this term "*auger*" because it properly describes this part of the tool; and as the "*drilling*" is performed by a *rotary motion* and moreover by a *true auger*, the process is properly described as a *boring* and not a drilling operation. For this reason I have referred to these machines as *boring* tools.

I am not aware that any comparative experiments have been made to determine the excess of fine coal produced by explosives of this class, and as such experiments would furnish results holding good only for coal of the same physical character, thickness, and hardness, lying at the same angle of dip, no general conclusions of value could be deduced from them. It has already been abundantly demonstrated by experience that the loss from the percentage of fine coal produced by the higher explosives is so large that it more than counterbalances the saving from increased efficiency.

We therefore find the use of explosives of this class restricted to sinking or driving, and to mining in workings where, from the presence of gas, the use of explosives proproducing a flame is extremely dangerous.

The details of charging and firing the shots are similar in almost every respect to the practice in other mining and quarrying districts. The charge is made up in the form of a paper cartridge, slightly smaller than the diameter of the hole, the hole is cleaned with the *scraper*, the cartridge is inserted, the blasting *needle* inserted, the hole is tamped, the needle withdrawn, the fuse (squib) adjusted and the shot fired.

Before firing a shot, the miners or laborers working in close proximity to the working in which the blast is located, are (or should be) warned by the miner, so that they may retreat to a place of safety.

From the Inspectors of Mines reports, it appears that over 15,000,000 pounds of powder were required to mine 27,000,000 tons of coal, or more than 600,000 kegs of 25 ℔s. each, or about 7,000 tons,—an average of about 9¼ ounces of powder for each ton of coal. The amount actually used varies from three or four ounces or less to one pound per ton.

Proper deductions must be made for coal mined by the men without powder, and for powder used in driving tunnels, gangways, and airways, in sinking slopes and other rock work.

In addition to the coal actually blown down, a considerable quantity is often left so shattered by the shot that it may be loosened and pulled down with a pick, crowbar, or or drill.

Picks are also used in cleaning the coal for loading, to split off layers of slate or bony coal which are found adhering to the larger lumps.

The common straight pick, one of the many forms of which is shown by Fig. 42, used for under-cutting in bituminous mines, also has many friends in the anthracite regions. Curved picks are also used, and are much preferred by some miners.

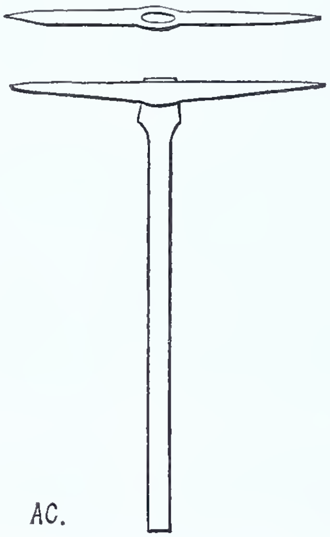

Fig. 42.

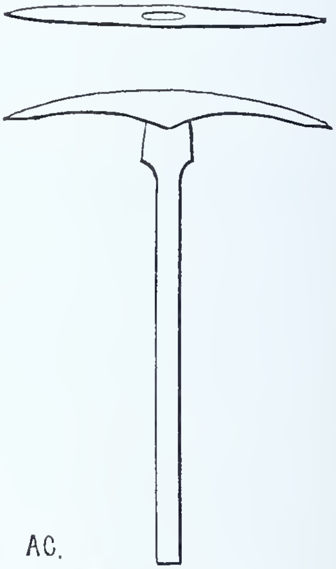

Fig. 43.

In cleaning the coal in the mine, generally on the breaker platform, picks with a single-curved point and a hammer-head, are often used. Picks of this type may be considered as a combined pick and sledge, and must be classed with combination tools, towards which few good mechanics are favorably disposed; but in work of this class this tool is very serviceable.

CHAPTER XI.

Underground Railways and Slopes.

It is sometimes assumed that the construction of a mine railway is similar in every respect to that of a road located on the surface, and, therefore, that a mine road may be constructed and kept in as good condition as a surface railway,—but it can easily be shown that this view is entirely untenable.

As the grade of the road is necessarily governed by that of the gangway, it is not always possible to adopt the grade best adapted for haulage of both loaded and empty cars; but as the gangway grade is commonly just sufficient for free drainage, the fall of the mine roads is nearly always *with* the direction taken by the loaded cars.

This fall averages from four to six inches per one hundred feet; the grade most likely to give the best haulage results, is from seven to ten inches per one hundred feet, or nearly double the grade commonly adopted.

As the haulage cost is small as compared to other items, this difference is so slight that it may be entirely ignored, and the gangway grades necessitated by drainage requirements may be considered to represent also a grade that may be properly adopted on all main-haulage roads.

The rough uneven floor of the gangway is the surface on which the sills are laid, and the top line of sills resting on knolls or ridges in the floor, determines the level of the road. The sills lying in depressions are blocked up to the proper level by placing beneath them a few pieces of slate, but it is not uncommon to see such sills *suspended by the rails.*

It is evident that a road built in this way, with the sills resting partly on rock or coal, and partly on loose slate, *cannot* be kept in good condition. Such a road-bed has none of the characteristics of a surface road in which the sills rest on earth, or rock ballast, which gradually yields until the sills have settled to a firm, uniform bearing.

The shocks to which the road is subjected from the cars, and the thrusts from the mules' hoofs (to which the sills act like cleats on a steamboat gangway plank) keep the track in a very uneven condition, so that there are few roads with either good alignment or even grade, and the shifting, unstable character of such roads inevitably leads to bad joints and irregular curves.

In addition to these, several other conditions unfavorable to the maintainance of good roads exist in greater or less degree at all colleries, the principal one being the rapid decay of the sills.

Broken slate or other mine refuse is used for ballast to fill in between the sills, and the whole track soon fills up with small fragments of coal and dirt; but as this is of a loose, shifting nature, and is constantly shaken up by the mules, it is of little service as ballast, and hardly deserves the name.

Branch gangways (in flat workings) are sometimes turned off at a very sharp curvature to the main haulage road, and the same may be said of branches running into underground tunnels, but as a rule the curves on main haulage roads are not excessively sharp.

Guage.

The same variance of opinion found among mine superintendents and engineers in every other mining district of the civilized world in regard to the best guage for haulage roads, exists to-day in the Anthracite coal fields.

It is generally admitted that a guage of more than four feet cannot be economically adopted, and that guages less than two feet and six inches are inadmissible. The guages most commonly in use are two feet six inches, two feet nine inches, three feet, three feet six inches, three feet nine inches, and four feet, but other intermediate guages have also been used.

Advocates of the broad guages believe that the greater stability of the track and the consequent reduction in haulage expenses, the increased capacity of the broad guage mine cars, the reduction in the outlay for rolling stock, and for repairs to the same, more than equal the disadvantages of broad as compared to the narrow guages.

Advocates of the narrow guages think that the ease of hauling around sharp curves, the reduction in cost of construction, and the use of mine cars with inside wheels, are advantages greater than those advanced by the broad guage adherents.

The maximum size of mine car that can be economically used is limited in its dimensions, (length, breadth, and height,) thus,—the length is limited to about ten feet, because any increase beyond this necessitates a broader wheel base to insure stability, the height is limited by the height of gangway and several other requirements, and the breadth is limited by the width of the gangway and turnouts.

It is certainly advantageous in some cases to decrease the height or length of a car and increase its width, and the broad-guage advocates have always advanced this as one of their cogent arguments in favor of the broader guages. But since the width of the car may be increased and the guage narrowed by building the cars with inside wheels (wheels inside the bed-frame) and as cars so built run more smoothly than the outside wheel cars of the broad guages, the whole controversy eventually concerns the relative merits of inside and outside wheels. This will be considered in detail in the chapter on rolling stock.

My own preference is for the narrow guages and cars built with inside wheels. Cars holding from ninety to one hundred and twenty cubic feet are built with inside wheels to run on guages of from two feet nine inches to three feet and three inches, and the results seem to be more satisfactory than when the guage is increased to four feet and the wheels placed outside.

Nearly all of the Philadelphia and Reading Coal and Iron

Company's methods of working shown by the plates reproduced from Report A² represent the guage as four feet and the cars are shown as outside wheelers, but this was simply a guage adopted for the illustrations, and is not to be taken as a type of all their operations.

In the Wyoming district the narrow guages have a firmer footing, and inside wheelers are more common.

Sills.

The sleepers used on inside haulage roads vary from the size of a regular surface railway sill down to sticks not much larger than a man's arm.

They are generally from four to six inches thick and from five to eight or nine inches wide—in other words they are usually made of round stuff six to nine inches in diameter hewed flat to a thickness of four to six inches.

At some collieries round stuff is used with two notches cut at the proper places to seat the rail, or the sill may only be hewed flat on one side, but these plans are seldom adopted at large collieries.

Their length is from four to six feet, depending on the guage of the road.

Switches.

The switch or "*latch*" most commonly used is shown by Fig. 31, in which a movable bar "*b*" is shown replacing the frog commonly used.

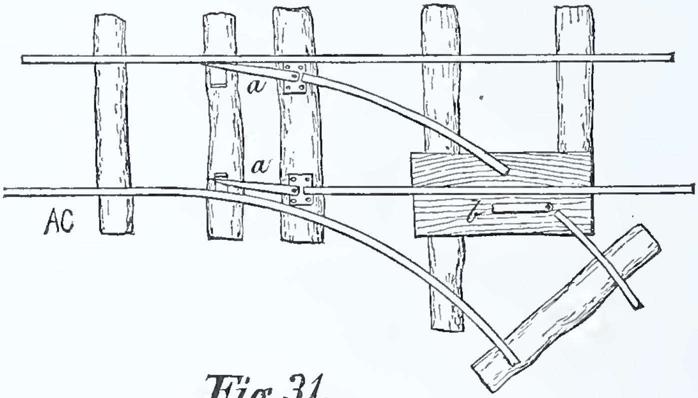

Fig. 31.

When the branch or siding is in constant use an ordinary railway frog is substituted for the bar "*b*." The latches *a*, *a*, are wedge-shaped bars of iron (made as high as the rail) with an eye in the thick end.

They are sometimes connected together by a rod attached to a lever so that they may both be moved at once from the side of the track or by a person situated at some distance.

This switch is made self-closing or automatic whenever it is necessary to run all the cars off at the branch, or to let all of them pass the branch (the switch then being used only to admit cars to the main track) by attaching the latches through a bar or lever to a metallic spring, a stick of some elastic wood, or a counter weight, to pull them back into a certain position whenever they have been pushed to one side or the other by the passage of a car on the main track. Figs. 35, 36, 38, and 39 show some of the applications of these spring latches or automatic switches.

By a similar arrangement, in which *two* counterpoise weights are used, the car may be made to set the latches, and sometimes the car is depended upon to effect this without the assistance of counterweights. (See Figs. 37 and 39.)

The counterweights used for this purpose are usually set on a rocking arm or lever oscillating a short distance beyond the vertical on one side, and through a greater arc on the opposite side of the center. The car opens the switch a certain distance moving the weight over the center; the weight then falls, carrying the switch over, and setting the latch firmly against the rail.

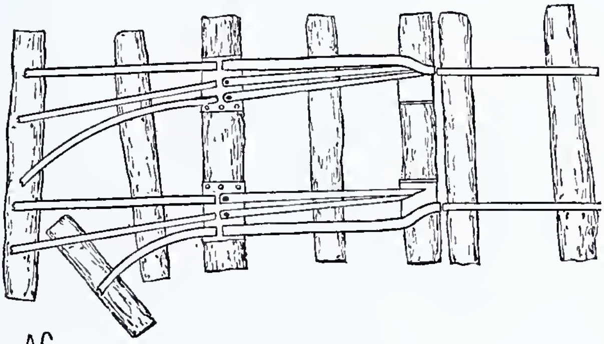

Fig. 32.

A modification of this switch is shown by Fig. 32, which represents a form of double switch largely used at the Lehigh Coal and Navigation company's collieries. These latches are set by the drivers, who kick them over and drop a small square of plate-iron between them to hold them in place.

This switch costs more than the other style, and is better adapted to outside roads than to the mine roads inside.

The ordinary movable rail switch in common use on all surface railways is also used to some extent in anthracite mining. It is commonly used on slopes arranged as shown by Fig. 37, to replace the "latches set by the car," and is also largely used on the mine roads at the surface.

Ordinary railway frogs and grade crossings are used throughout the region, but the grade crossing is sometimes replaced by a small turntable, which then answers two purposes, and sometimes by the arrangement shown by Fig. 33, in which four movable bars are thrown across the main track whenever the other road is to be used.

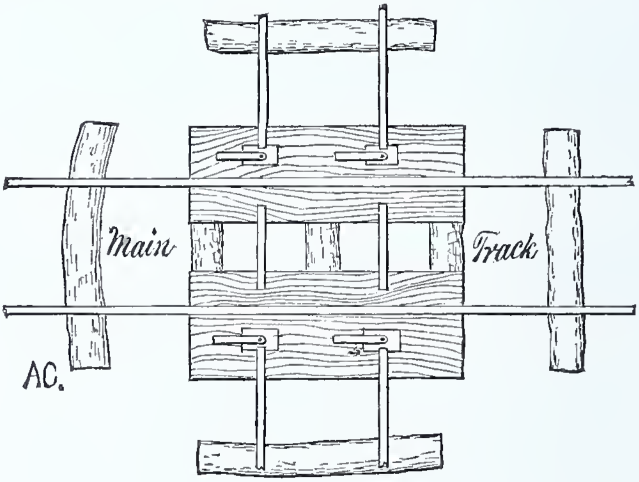

Fig. 33.

The subordinate road is built about one and a half to two inches higher than the main road to allow the bars to clear the main track rails.

Rails.

Wooden rails are not used on main haulage roads. In buggy breasts driven up rather steep pitches wooden rails are sometimes used, but even in such cases the track is often of iron and wood combined.

On steep gangway roads a wooden tram is sometimes laid against the rail to increase the friction and enable the runner to control the descent of the car.

Strap-iron rails have been almost entirely replaced by the forms shown by Fig. 34. In this, as in many other respects, mine railroading has availed itself of the improvements effected in surface railroading, and the heavier rails now used are identical in form with rails now laid on some of the narrow-gauge steam railways.

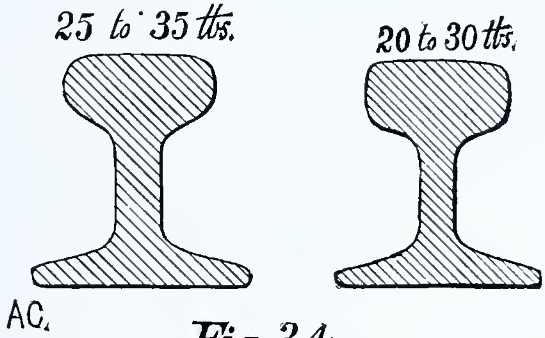

Fig. 34.

The joints and fish-plates are often the same with those used by narrow-gauge roads, but on many mine roads, especially those not used as main haulage roads, no fish-plates are used, the end of the rail being held in place by two or three spikes driven into a sill lying beneath the joint, or the joint is made by a shoe or chair which receives the ends of both rails.

Turnouts.

On gangways used as main haulage roads it is usual to place turn-outs at certain intervals to allow the loaded and empty "trips" (trains) to pass. These turnouts are commonly constructed by widening out the gangway to the required width for two tracks, and for a sufficient distance to accommodate from five or six up to fifteen or twenty cars. The switches at each end may be made self-acting so that the empty trip coming in is thrown on the turnout, and in running out on the main track at the other end the cars open the switch, which immediately closes.

But as there is constant trouble with self-setting switches, either from small fragments of coal or slate clogging them up, or from insufficient power of the spring to move them, they are viewed with disfavor by many engineers, who do not care to use them under any conditions.

Slopes.

At the foot of a slope, or at the landing on any lift, the gangway is widened out to accommodate at least two tracks—one for the empty and one for the loaded cars.

The empty track is generally on the upper side of the gangway, or that side nearest the floor of the bed, and the loaded track on that side of the gangway nearest the roof of the coal.

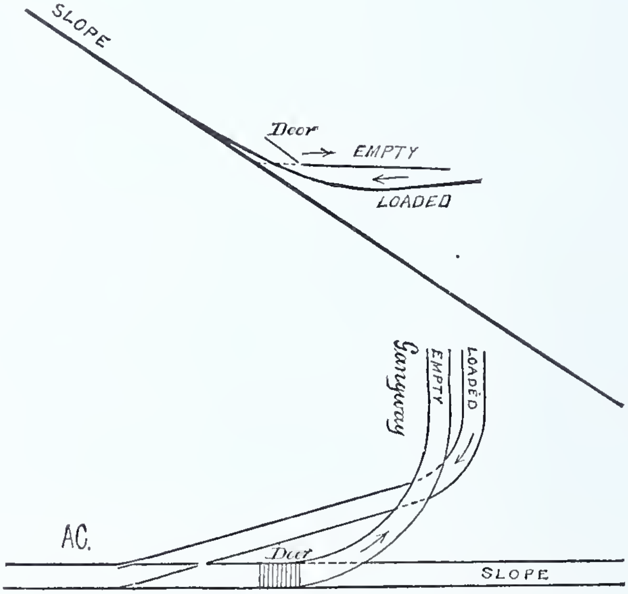

Fig. 35.

An arrangement in use in the Shenandoah and some other districts is shown by Fig. 35. At a distance of forty or fifty feet above the gangway the slope is widened out to accommodate the branch leading into the gangway loaded track. This branch descends with a gradually lessening inclination until nearly at the level of the gangway it turns into the main loaded track. A short distance above the gangway a bridge or door* is placed, which when closed forms a latch by which the empty cars are taken off the slope. The empty track is about six feet higher than the loaded track, and is carried over it on a trestle. The illustration shows the plan as arranged for a single slope, or one side only of a slope taking the coal from both directions.

When coal is being raised from this lift the bridge is closed; the empty car comes down and is run off over the bridge, the car is unhooked from the rope (chain) and the chain and hook are thrown down to the branch below on which a loaded car is standing; the loaded car is attached, the signal given, the car ascends to the main track on the slope, opening the switch,—or the switch may be set each time by the bottom-men by means of a lever at the bottom of the branch.

This plan can only be economically employed in thick seams, as the height necessary to allow one track to cross the other on a trestle, cannot be obtained in seams of moderate thickness without taking down a large amount of the top.

A more simple plan is in common use which dispenses with the bridge.† The branch is laid off as shown by the above figure, but near the point where it enters the gangway, a switch opening into the empty (a loaded) track is placed.

It will be observed that in this arrangement the tracks cannot be as well arranged for handling the cars by gravity as in the former plan, in which the empty cars when detached from the rope run by gravity into the empty siding, and the loaded cars descend by gravity around the curve to the foot of the branch, where they lie ready to be attached to the rope.

When the pitch of the bed is so steep that the coal falls out of the cars, the coal is raised in a gunboat, or the cars are raised on a slope carriage,—in either case the arrangement of the tracks at lift landings is entirely different.

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*This door carries two tracks that form hinged latches similar to those shown by the Atlas Sheet illustration of the truck-pit at the Diamond Breaker.

†When the bridge is raised the track is open and coal may be raised from a lower lift.

With either a gunboat or slope carriage the arrangement of tracks on the slope is the same, but in the former case a connection between the slope and the gangway tracks is often advisable.

When a gunboat is used the gangway tracks run direct to the slope and a tipple (dump) is placed on each side to dump the mine cars over the gunboat; but when the cars are raised on a slope carriage the gangway tracks run direct (at right angles) to the slope to carry the car to the cage.

The bottom arrangements are similar in many respects to the lift landings, but the slope is continued down a few feet to make a pit to receive the carriage or gunboat while being loaded.

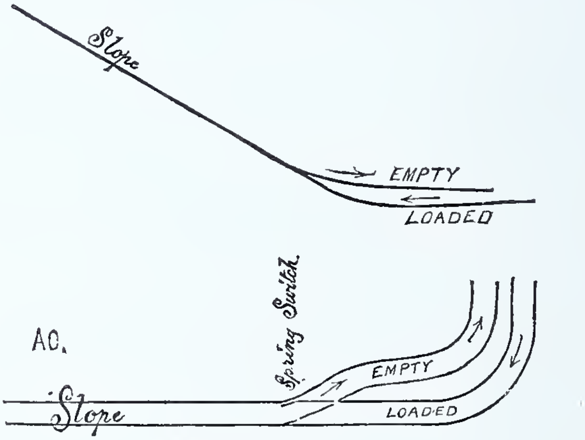

Fig. 36.

A common arrangement of the tracks at the bottom of a slope is shown by Fig. 36. A branch is made by widening the slope out near the bottom, and this being a few feet higher than the main track is used to run off the empties by gravity. The loaded cars run in by gravity around the curve to the foot of the slope in position to be attached to the rope.

In ascending, the loaded car forces its way through the switch, or the switch may be set by a lever located at the foot of the slope. When the empty car descends, it runs in on the branch, where the chain is unhooked and thrown over in front of the loaded car, and runs around the curve into the gangway by gravity.

It will be observed that in this plan the loaded car (and consequently the bottom-men) stands on the track *in line with the slope*, and in danger from any objects falling down the slope, such as fragments of coal, etc., or from the breakage of the rope or couplings, but this can be obviated by making the bottom on the curve. The illustration shows only one half of the slope, the other half is of course similar.

All of these plans necessitate the location of that part of the gangway near the slope, in the upper benches of the coal, or near the top-rock,—the gangway is then curved gently around toward the floor so that when it has been driven far enough to leave a sufficiently thick pillar, the bottom bench is reached and the gangway is then driven along the bottom-rock.

A very different bottom arrangement is shown by Fig. 37, which also represents a plan frequently adopted on dirt planes.

The two slope tracks are merged into one a short distance from the bottom of the slope, and on the opposite side of the bottom, two tracks curve around into the gangway on opposite sides of the slopes. As these branches curve into the main gangway tracks, a switch sends off a side track for the empty cars.

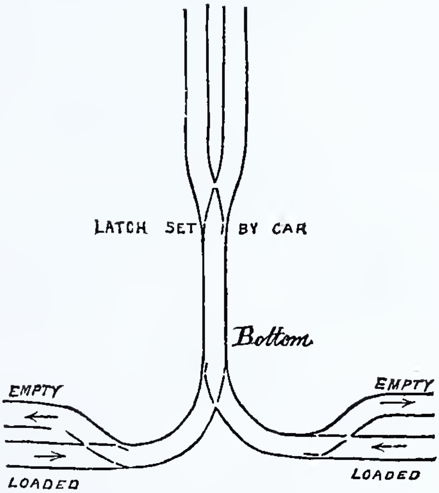

Fig. 37.

The switch on the slope is either set by the car,—and this can be done because the next loaded goes up on the same side on which the last empty descended—or by a lever located at the bottom.

It will at once be seen that in this plan no opportunity is afforded of handling the cars by gravity. The curved branches are made nearly level and the momentum of the descending car, if quickly detached, is often sufficient to carry it partly or wholly around the curve, even against a slight adverse grade. The disadvantage above noted of having the bottom in direct line with the slope (where there is danger from breakage and falling coal) also obtains to this plan.

In the plan shown by Fig. 38 the grades may be so arranged that the cars can be entirely handled by gravity.

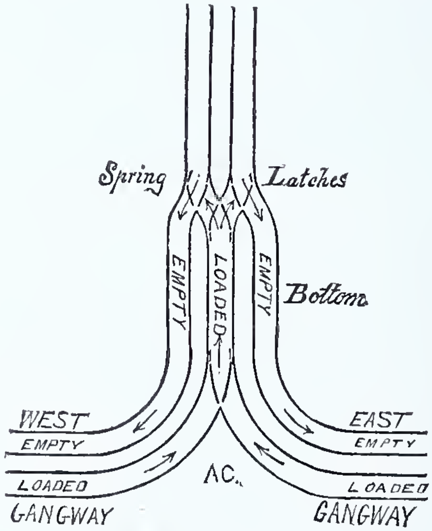

Fig. 38.

The latches on the main slope track may be closed automatically by a spring or weight, the loaded car running through them in its ascent on the slope, or both sets may be operated by a single lever at the bottom. The switch at the upper end of the central track ("loaded") is set by a hand lever. All three sets may be linked together, so that they can all be properly set by a single lever.

Reference to Fig. 36 will show that this is only a modification of that method. It requires space at the bottom for only three tracks, while the former requires width to accommodate four tracks, but is objectionable because it is more complicated. The extra set of latches at the top of the central track, and the curvature of both main tracks into this central one, must inevitably cause much trouble and delay from cars jumping the track at this point.

The plan shown by Fig. 39 is open to many of the objections pertaining to some of those already described, and which need not be reiterated here. It can only be employed in thick seams, or in seams of moderate thickness lying at a slight angle of dip.

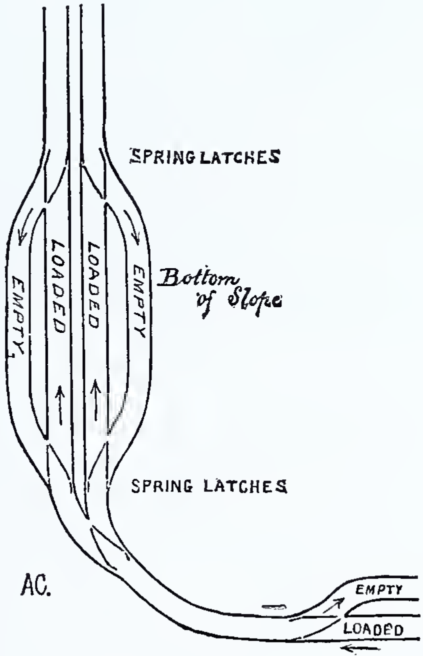

Fig. 39.

In planning the arrangement of tracks on a slope, it is considered advisable to place as few switches as possible on the slope itself ; to keep the main track unbroken ; to make the tracks as straight as possible ; to have nothing standing at the bottom in direct line with the slope tracks ; and to arrange the tracks so that the cars are handled by gravity.

The arrangement of tracks near the top of the slope, and on the surface, is often very similar to the bottom arrangements above described ; but as all the loaded cars[*] are to be sent off on one track, and all the empties come in on the same track to the head of the slope, and as there is usually abundance of room for tracks and sidings, these top arrangements are in a measure much more easily designed.

At some collieries the two main slope tracks run into a single track near the head of the slope,—a plan somewhat similar to the bottom arrangement shown by Fig. 37,—and the cars are then all brought to the surface on one track, which, after passing the knuckle, bifurcates into a loaded and empty track.

A similar arrangement is frequently adopted at slopes on which a carriage or gunboat is used.

When the two main slope tracks are continued up over the knuckle to the surface,—the most common and the best plan,—arrangement of tracks and switches may be planned entirely with a view to the quickest and most economical method of handling the cars, but a consideration of this belongs to another part of this report.

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[*] Except the cars containing rock or slate, which are taken off on a switch from the track on which the loaded cars stand.

CHAPTER XII.

Slopes, Planes, and Inside Slopes.

The arrangement of tracks and switches on slopes has already been described. Precisely similar arrangements are used on inside slopes, and on planes, whether above-ground or inside the mine.

Slopes used as main outlets are almost invariably provided with double tracks, so that the loaded car may, in part, be balanced by the weight of the empty car, but the same object is sometimes accomplished by laying three rails with a turn-out midway between the top and bottom. This latter arrangement has been considered in the chapter on Slope Sinking and Timbering.

When the topography of the surface near the slope mouth permits, and when the mouth of the slope is not at too great a height above the railroad by which the coal is to be shipped, the prevailing practice is to build the breaker directly in front of the slope and to continue the slope tracks up on an open trestle to the top of the breaker. When the mouth of the slope is considerably higher than the railroad, the breaker is located so that the main dump (at the top of the breaker) is slightly lower than the mouth of the slope.

When the former arrangement is adopted, the winding engine is frequently located inside of the breaker,—a plan that has already been discussed in considering the location of the various structures that make up a colliery plant,—but this plan is rapidly falling into disfavor, and in building new plants a brick or stone engine-house is now commonly erected for the winding and pumping engines.

At collieries having a small output, where speed in hoisting is not important, the winding engine is frequently also made to do duty as a pumping engine. This is accomplished by operating the drum by clutches or friction gearing,—the engine running constantly.

When a large output is to be raised, the winding engine is always independent of the pumps.

To secure sufficient lead between the knuckle pulleys and the drum, the engine is placed at some distance from the mouth of the slope, and is so located that the cars on one side may either pass *over* or *beneath* the rope on the opposite side of the slope.

When the winding engine and drum are placed below the knuckle, the rope is sometimes passed through a slide working like a cross-head between guides ten or fifteen feet long. This slide is attached to a counter-weight, and the hole through which the rope plays, although large enough to pass the rope freely, will not pass the rope socket, the hook, or a stop placed on the chain. After the car is detached the drum is turned back one third or one quarter of a revolution, the counter weight draws up on the rope keeping it taut upon the drum and pulling it out to within a few feet of the knuckle, where the empty car is to be attached.

When the slope tracks run directly up to the head of the breaker the coal is either raised in a gunboat or self-dumping skip, or in a mine car of ordinary construction.

When the coal is raised in a mine car the car is sometimes unloaded from the rear end. When this is done the slope tracks are continued up a few feet above the main screen bars, the car is raised with the door at the lower end, and when vertically above the shute or screen bars the door latches are knocked up and the coal slides out.

A car-dumping barney will be described in another part of this report, by which the cars are dumped from the front end, by running the front wheels into a chair and raising the rear end of the car.

That there are serious objections to raising coal in cars with the door at the lower end is evident; but the disadvantages and expense of handling the cars at the head of the breaker by the old plan are so great that any other system seems preferable.

At quite a large number of slope collieries at which the cars are raised directly to the top of the breaker, the cars are detached at the knuckle, run some distance to the dump to be tipped, switched off on an empty track, and run in again on the main track to the knuckle before being lowered. When the output is large, this arrangement requires from six to eight men and boys at the head of the breaker to handle and dump the cars; while the same work can be done by two or three men when the cars are dumped without being detached from the rope.

Planes.

The term "*plane*" is applied to any inclined plane on the surface, and to inclined planes built on trestlework on which cars are raised from the surface to the top of a breaker; and this term is applied to such inclines whether they are used for lowering cars by gravity or for raising them by steam power.

But inside the mine workings the term *plane* is only applied to inclines down which coal is *lowered.*

Dirt planes are built and operated exactly like the self-acting planes of the bituminous region, except that, as the waste is to be raised to a height, they are operated by steam power.

When the amount of material to be raised is very large a separate hoisting engine is provided. This is often located a considerable distance below the top of the plane, and the ropes are carried up and passed around sheaves at the top. Steam is conveyed from the boiler house through a steam pipe laid for that purpose. By this arrangement the head of the plane can be raised (the plane lengthened) as the dirt bank increases in size, by simply moving the sheaves; while with the engine is located at the top it is necessary to rebuild the whole plant to lengthen the plane.

When the plane is short and only a moderate amount of dirt is to be raised, the drum is frequently operated by a shaft taking power from the breaker engine or from some main driving shaft in the breaker. If the plane is single the descent of the car is controlled by a brake, and only a single clutch is needed, but when the plane is double, double clutches or friction gearing is used to take power from the driving shaft.

Cone friction gearing has been largely used for this purpose.

Short planes are sometimes introduced between the breaker and the mouth of a shaft or slope to raise the cars a few feet so that they may be run by gravity in both directions. Such a plane is shown by Atlas Plate No. XVIII at the Hammond Breaker. The power is taken by gearing from the breaker shafting, and the plane is operated by an endless chain with flat links, provided at intervals with an upright catch that catches behind the front axle of the car and raises it to the top of the plane. Planes of this class usually run at a very low speed—one to two hundred feet per minute.

Inside planes.

Inside planes are self-acting planes, *i. e.*, the weight of the loaded car raises the empty car. In timbering and in the arrangement of their tracks they are similar to slopes; in other respects they resemble the inclined planes of the bituminous region.

Like inside slopes they have their upper terminus on a gangway, and a chamber excavated in the coal on the upper side of the gangway is fitted up to hold the drum.

Planes cannot be operated on a slope of less than five degrees, and are seldom seen on slopes exceeding thirty-five or forty degrees; but there is no good reason why they may not be operated on much steeper dips. But when the coal dips more than forty degrees the method of mining rarely affords favorable opportunities for the employment of this means of handling the coal.

Small inside planes are sometimes built by laying tracks in a worked-out breast, not for lowering coal, but for raising timber, cars, machinery, and other supplies, to workings opened by a counter gangway.

Inside slopes.

In the arrangement of the tracks and switches at landings on inside slopes, the same plans are adopted as on ordinary slopes, and in the method of operating they do not present any distinctive features.

As the top of such slopes always opens on a gangway which may be in use as a haulage road, it becomes necessary to devise some plan by which the continuity of the gangway tracks is not broken by the slope.

When the bed is thick the gangway may be driven around the mouth of the slope in the top benches, and the cars taken off the slope over a bridge. When the cars are raised on a slope carriage, a bridge is not necessary as the cars are simply run off on one side of the slope over a track connecting with the main gangway track,—or the cars are run off over a door closing in behind the slope carriage.

By starting the slope in the upper benches and sinking gradually through to the lower bench, enough space may be obtained at the upper gangway to run the main track *in front* of the knuckle, the cars on the gangway then passing beneath the rope.

When the coal is soft, and large excavations are not safe, the gangway may be continued around the head of the slope as an independent passage cut on the upper benches.

The continuity of the gangway track is sometimes effected by a door (like a short draw-bridge) spanning the top of the slope, which is closed whenever a trip of cars passes.

Another plan is to continue the slope up a few feet above the gangway,—the gangway being driven at a sufficient distance from the floor to allow space for its tracks to cross in the coal, or over a staging just behind the slope mouth, —and to run the cars down over a landing branch similar to the "empty" track of Fig. 36.

When the cars are raised on a slope carriage, the slope may be made to open directly upon the gangway, and the continuity of the gangway track may be preserved by transfer platforms. (See Page plate No. 24.)

When the two slope tracks converge into a single track at the top of the slope, only one such platform is necessary.

A transfer platform in use at the Empire colliery is shown by Page plate No. 24. This platform carries rails which complete the continuity of the gangway track. The platform is provided with shoes or runners which take the place of wheels and rest upon the slope track. It is found in practice that platforms supported on wheels get loose and "wabble" about from the impact of cars running over them. When in use the platform lies on stops which support it at the level of the gangway. When the slope carriage rises it strikes the deadwood (buffers) of the transfer platform, and slides the whole platform up out of the way ; as soon as the car is taken off, an empty car is run upon the slope carriage, the latter descends, and the transfer platform slides down until checked by the stops.

A device of this kind is especially valuable in thin seams where sufficient room for other arrangements cannot be obtained without taking out a large amount of the roof or floor.

When the bed is thin and the cars are not raised on a slope carriage, the best plan is to continue the slope up a short distance above the gangway, and run the cars off on a curved branch, or over a bridge.

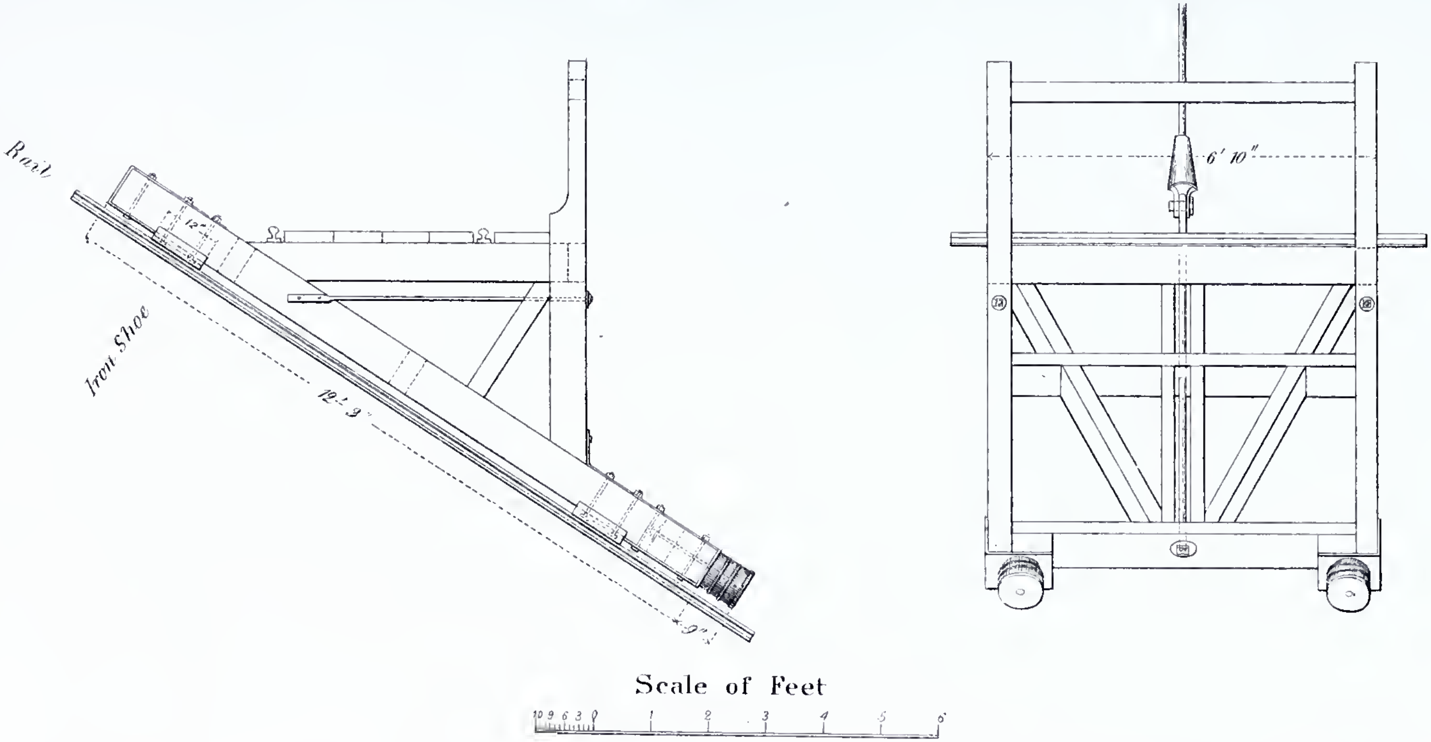
The most important problem in the operation of inside slopes is that of motive power. As it is especially important (whether steam or compressed air is used) to reduce the power necessary to operate to a minimum, it seems advisable to equip all such slopes with two ropes, so that the resistance to be overcome will only be that due to the weight of the rope and coal, and friction, which at the middle of the lift will be reduced to the weight of the coal plus friction, and in the last half of the lift, less than this by the excess in weight of the rope on the empty side of the slope.

As opposed to this we have on single-track slopes the weight of the car, the coal, the rope, and resistance due to friction throughout the whole lift.

But single-rope slopes are much more common than we might *a priori* expect to find them, notwithstanding the disinclination to the use of steam underground.

The use of steam underground is viewed with such dis-

SLOPE TRANSFER CARRIAGE AT THE EMPIRE COLLIERY.



favor that a large number of underground slopes are now operated by engines located at the surface; the rope being conveyed through a pump-slope, breast, travelling-way, or airway, or air-shaft opening at the surface, or even through a bore hole.

A bore hole six inches in diameter and about six hundred feet deep was lately drilled at the Hollenback colliery to be used for this purpose.

When the power is supplied by an engine located at the surface this objection to the use of a single rope disappears.

At the Wyoming colliery, near Wilkes-Barre, two inside slopes are thus operated through the air shaft.

When the power is obtained from an engine run by steam inside the mine, the disposal of the exhaust steam is a problem similar in every respect to that incident to the employment of steam pumps underground.

In addition to trouble from the exhaust, there is necessarily more or less loss from radiation and condensation.

A partial solution of this problem has been effected at some collieries by the substitution of compressed air for steam, not only for the slope engines but also to run the pumps.

At collieries in which there is a considerable amount of rock-work, (tunnel driving, etc.,) which may be more cheaply and quickly done with power drills, there can be no question as to the expediency of erecting air-compressing machinery to supply the place of steam at the inside pumps and slope engines; but when no work of this class is needed, and there is perhaps only one inside slope engine, the cost of erecting and maintaining a compressing plant would doubtless be greater than the saving effected by increased life of mine timber, etc.

This subject is now receiving much attention from mine operators and engineers. It must be thoroughly studied in all its bearings and in the light of carefully prepared statistics before it will be possible to determine in advance whether, in the case of any particular working, compressed air can be advantageously substituted for steam.

Underground slope engines and drums do not differ from those in use on the surface, and at a large number of inside slopes the winding plant consists of old machinery that has been used at the surface.

As a chamber must be excavated in the coal above the gangway to hold the winding machinery, and as it is often difficult and never advisable to make this engine-room very large, large drums are seldom used.

The engines are generally small and geared to the drum by second or third motion. As the output from any one inside slope is seldom very great, the necessity for fast winding and first motion engines does not exist.

The engine and drum are often bolted to a frame of heavy sills, held in place by being notched and wedged into the rib on each side of the engine-room, and in some cases sunk in trenches hewed in the floor.

Atlas Sheet No. IX illustrates a dirt plane designed to raise cars from two different levels at Breaker No. 10 of the Lehigh Coal and Navigation Company. The engine is located a short distance above the upper truck-pit, and the ropes are carried up on bearing pulleys to horizontal sheaves at the head of the plane.

The ropes are wound on two drums placed side by side on the same shaft. The diameters of these drums bear the same ratio with the lengths of lift on each side of the plane, so that when a car has been raised to the top on the long lift, the barney on the opposite side has descended into its pit, and *vice versa.*

This drawing also shows the arrangement commonly used when the cars are raised by a barney. The barney (ram or truck) is a small car running on rails inside of the main track, which descend into a pit at the bottom, so that the car may be run in *over the barney* to the foot of the plane.

Hinged latches (bridges) are shown at the landings above both of the barney-pits, by which the empty cars are run off the plane at a sufficient vertical distance above the foot of the plane to allow the empty cars to be run to the breaker and return (loaded) to the foot, by gravity. When a car is being raised these latches are lifted to allow the cars to pass beneath them.

Plans similar to this might be used on slopes for raising coal from two different levels at the same time, but the adoption of such a complicated system is open to many objections.

Prominent among these is the fact that such a winding plant resembles a compound windlass, and if one side of the plane is very much greater than the other, the weight of *the empty car* and rope on the long side *will raise the loaded car* through the upper part of the lift.

It is evident that when it becomes necessary to lengthen either side of such a plane, the relative diameters of the drums must be altered.

Difficulty will also doubtless be experienced from unequal wear of the lagging on the two drums, altering the ratio between their diameters.

Gravity plane.

A gravity plane fitted with horizontal sheaves (instead of a drum) is shown by Atlas Sheet No. X. This plane is located at the Lehigh Coal and Navigation company's No. 8 colliery.

As the drawing clearly shows the details of this plan, it is not necessary to give any further explanation of its construction.

It will be observed that the rope after turning around the large sheave runs over the smaller sheave and back to the main sheave, making two turns; that the large sheave is controlled by a brake, and that the rope passes through counterweighted slides (stops) which produce sufficient tension to prevent the rope from slipping after the car is detached at the top.

The advantages of this method of replacing the drum and two ropes by sheaves and a single rope will be considered in the chapter on winding machinery.

In the arrangement of tracks this plane differs from any of the plans described in other portions of this report. It consists of four rails to within a few feet of the bottom, where a central rail replaces the two inside rails, and a few feet below this the three rails are merged into a single track. By reference to Fig. 37 it will be seen that this is simply a modification of that plan.

CHAPTER XIII.

Rolling Stock and Motive Power.

Mine cars of a size almost unknown in other mining districts are in use throughout the anthracite regions. In English mines the trams or mine cars rarely hold forty or fifty cubic feet, and in the coal-mining districts of Continental Europe larger mine cars are rarely seen.

In American bituminous mining practice larger cars are used, but cars even approaching in size to some in use in the anthracite regions could be taken into very few bituminous mines.

The present large size of anthracite mine cars is due to the great thickness of the seams now being worked, and to the ready adaptation by our mining engineers of the knowledge, gained by the long and dearly-bought experience of railroad managers, that freight can be more cheaply loaded, transported, and handled in large than in small cars.

Several conditions, without which such large cars could not be economically used, have favored to an unusual degree the use of very large cars:

1. Thickness of the beds worked.

2. Steep dip of the beds allowing the cars to be loaded by gravity; and increasing the possible height of the gangways in seams of moderate thickness.

3. Gaseous condition of the mines necessitating large gangways for thorough ventilation.

In *height* the cars are necessarily limited by the height of the gangways; and in flat workings by the thickness of the bed.

In *width* they are limited by the width of the gangways.

In *length* by the wheel base and by the sharpness of the mine road curves.

The length seldom exceeds ten feet, the width five feet, and the height (from the rail) five feet and a half.

The cubic contents of cars commonly used ranges from seventy to one hundred and forty cubic feet, or in weight from one ton and three quarters to four tons. These are the extremes,—a mine car of average size holds about one hundred to one hundred and ten cubic feet, or in weight from two and a half to three tons of coal when loaded full.

In the Lehigh district very large mine cars are more common than in the Wyoming Basin. This difference is caused by the prevalence of steep dips in the former district.

The Hazleton district can boast of some very large cars, as, for instance, those in use at the No. 6 colliery and other workings of Pardee & Co.

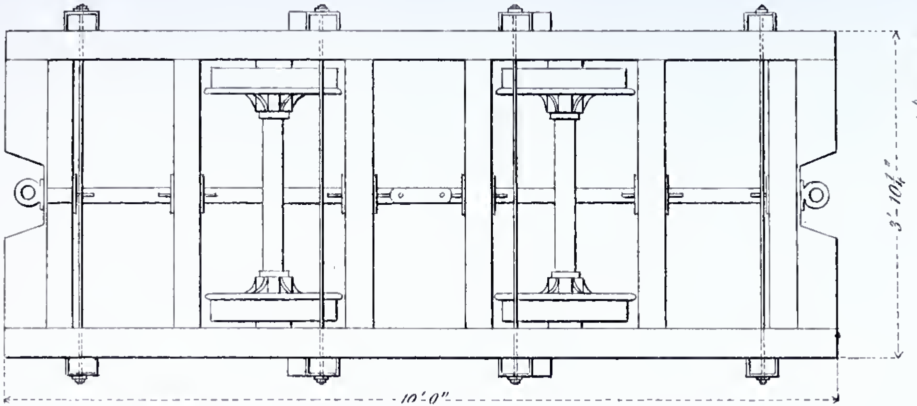
At the Lehigh Valley Coal company's collieries, near Shenandoah, cars of one hundred and twenty to one hundred and forty cubic feet capacity are used. At collieries Nos. 2 and 4 cars of one hundred and forty cubic feet, carrying from three and three quarters to four and a half tons are used ; at the No. 3 colliery the cars hold one hundred and thirty-three cubic feet, and at the No. 1 colliery the cars carry about three and a quarter tons, holding one hundred and twenty cubic feet. These cars are estimated by Col. Brown to weigh, empty, about 4500 and4757 ℔s.

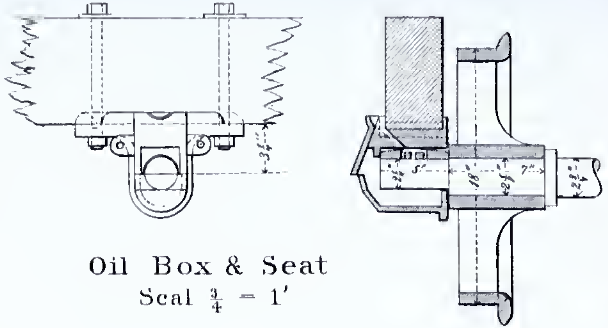
In shape and in the methods of construction the mine cars in use present an almost infinite variety. Some are constructed almost entirely of wood, others entirely of iron, but the most common plan is to use wood braced with iron tie rods or bolts for the frame, and to protect and stiffen the joints by angle iron and cast-iron shoes.

In shape they vary from the broad low car of some of the Wilkes-Barre mines to the high narrow car used in the Schuylkill region, and from the square upright pattern to the hopper-shaped style.

In thin and flat seams the broad, low car is preferred ; the hopper-shaped car is best adapted to the narrower guages when it is desired to increase the capacity without increasing the length or height of the car

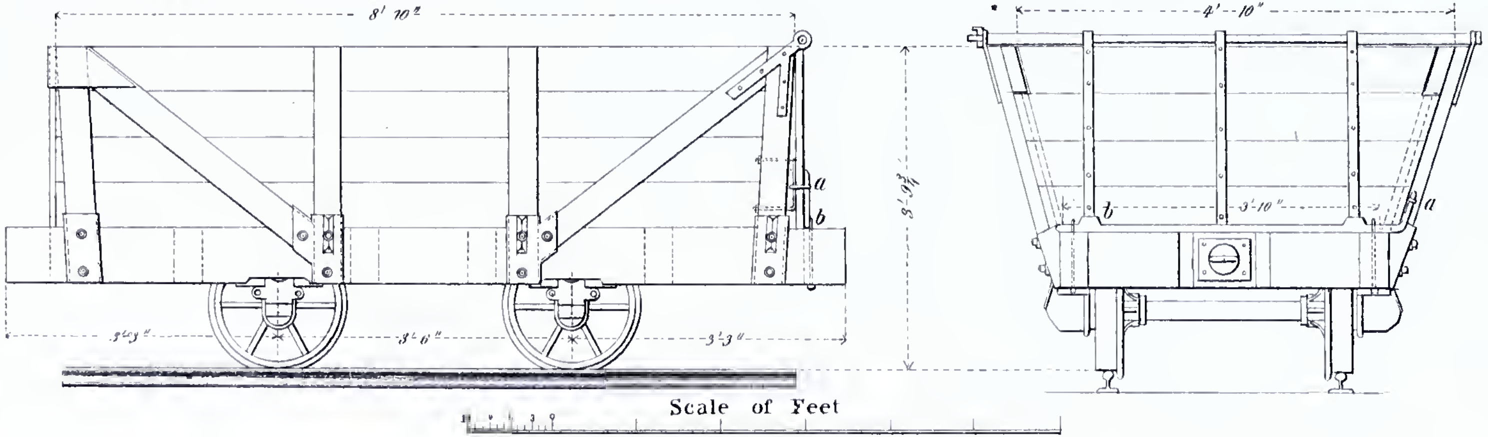
Large square cars of moderate height with a capacity of one hundred cubic feet or more, are slowly but surely replacing all other patterns at mines where the height of the





Oil Box & Seat
Scal ¾ = 1'

MINE CAR AT HOLLENBACK COLLIERY L. & W. COAL COMPANY.



seam, etc., permits; and the present tendency is strongly in favor of inside wheels.

They are supplied with a swinging door hung on an iron rod which runs across the top of the car through both hinges. The doors are locked by a double iron latch catching in a lip or latch holder on each side of the car, as in Page Plates 26 and 29, by a single vertical bolt or swinging latch in the center of the car just over the draw-hook (Page Plates Nos. 27 and 28)sometimes operated by a lever at the side of the car, by a rocking shaft carrying lips which catch the bottom of the door, (see Plate No. 25,) or by a movable iron hook (latch) placed bottom side up (to work by gravity) on the side of the car. Other fastenings have been used, but the double iron latch is the style now generally preferred. Complete sheet iron lining is frequently used, but it is not always considered necessary, as the car frame will often wear out as fast as the lining. The floor of the car is lined with iron plates about one eighth of an inch thick, which not only protects the plank from damage, but is of considerable advantage in allowing the coal to quickly slide out when the car is dumped.

Cars constructed of boiler iron have been used at a number of collieries, especially in the Lackawanna district, but as they have not been adopted at recently opened collieries equipped with the most perfect mining plants, we may conclude there are important objections to their use that outweigh any apparent advantages they possess. We may indicate some of the disadvantages: 1, Cost; 2, Difficulty and expense of repairs; 3, Their rigidity increases the wear on the wheels, axles, and track; 4, Their weight,—*for equal strength.*

Wheels and axles.

Cast-iron wheels are almost always used. Cars have been built with wrought iron spoke wheels, (see Page Plate No. 26. Summit Hill car.) but the ordinary cast-iron wheel seems better suited to mine work, and is much cheaper.

The most important problem which must be solved before the best form of mine car can be determined involves the relative merits of the inside (or fixed) and the outside (or loose) wheels.

Viewed from the stand-point of a mechanical engineer, there can be no doubt of the superiority of the inside, over the outside, or loose wheel. The inside wheel cars run smoothly, while nothing can be more uncertain than the irregular running of loose wheeled cars. In the language of a prominent engineer, "We cannot get any two loose-wheeled cars to run alike."

On the other hand, the loose-wheeled cars are *cheaper*, they are more easily lubricated, broken wheels can be instantly replaced by new ones, and the axles are less likely to bend.

I think the advantages of each style may render it superior to the other under certain conditions. At collieries hauling the coal a long distance underground on good roads, or transporting it in mine cars a considerable distance to the breaker, the inside wheels are doubtless the better form; but at mines transporting the coal only a short distance and on very bad roads, or over frequent sharp curves, outside wheels may give better results.

Loose or free running wheels are generally lubricated from the outside or through a tap hole in the hub, but several attempts have been made to obtain a self-lubricating wheel, by making either the axle or wheel carry the oil. The objections to hollow axles have prevented the adoption of this class of inventions, but self-lubricating wheels with hollow hubs have been successfully used at several collieries.

The difficulty of producing a wheel with hollow hub free from flaws, the increased weight of such wheels, their tendency to unequal wear, greater liability to breakage, and increased cost, have in the past and probably will in the future, prevent the general adoption of this method of lubrication.

Cars with loose outside wheels and *axles running in pedestals* do not seem to have any marked advantage over those with the simple loose wheel and fixed axle.

Almost as many different forms of axles and boxes as of mine cars are in use in the anthracite region, and it does

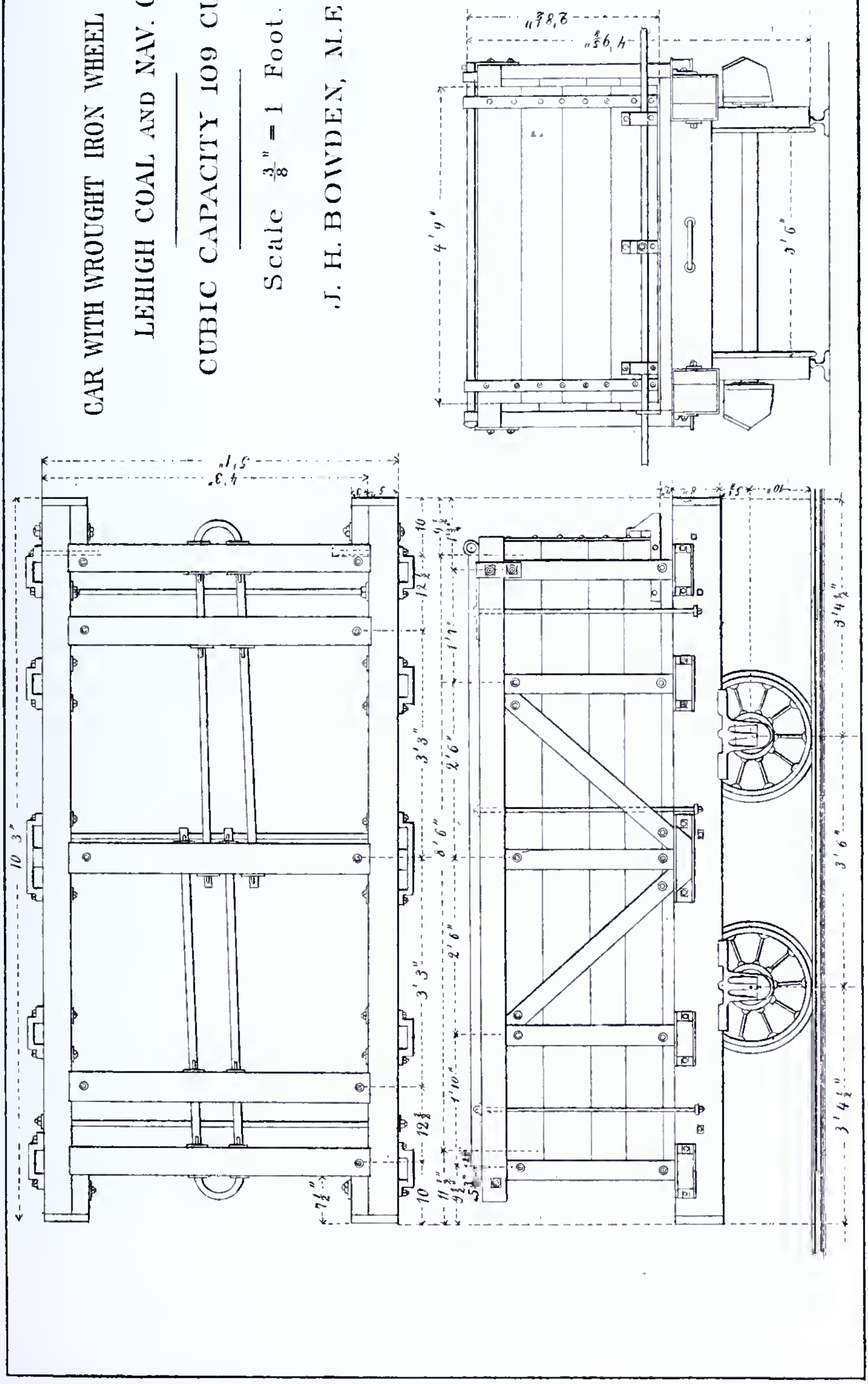
CAR WITH WROUGHT IRON WHEEL SPOKES

LEHIGH COAL AND NAV. CO.

CUBIC CAPACITY 109 CU. Ft.

Scale $\frac{3}{8}'' = 1$ Foot.

J. H. BOWDEN, M. E.



A. Winslow Del.

not seem possible to speak of any one style as being superior to all others. In this as in railroad practice, there are great differences of opinion. We may, however, indicate some of the requisites of a good oil-box and journal bearings.

Tightness is a prerequisite, and in dry mines where the dust is very penetrating, this is especially important; the bearings should be sufficiently broad; the oil-box large enough to hold enough oil to run a month without renewal; and so constructed that while it may be quickly and easily opened, it will not open by jarring or by being accidentally struck by a sprag or lump of coal.

The *draw-bars* or *draw-hooks* are very commonly run through the main sills of the car and held by a washer and nut. Sometimes they are run through all the sills, sometimes through only two. Another plan is to run the two bars beneath the sills and fasten them together, thus practically making one bar of the two. Various forms of draw-hooks are shown by the mine car page plates, but in all of these the strain is transmitted to the transverse sills instead of to the main sills. A better plan, which is used by the Philadelphia and Reading Coal and Iron company, is to split the bar into a Y, or make it double, and fasten each arm to the main sill at the side of the car.

In addition to the central draw-hook, a draw-hook is sometimes placed on each side of the car. When the car is to be raised on steep slopes, or on slopes on which the angle of pitch is not constant, these draw-hooks are pivoted, so that the car is not thrown from the track by a change in the slope angle.

The *deadwood*, *buffers* or *bumpers*, are usually placed at the side of the car in prolongation with the main sill.* They are simply blocks of wood, (surrounded by an iron band to keep the wood from splitting off,) from five to ten inches deep, five to eight inches high, and eight to twelve inches or more broad.

Other details of construction will be more clearly indicated by the following description of the mine cars illustrated by Page plates Nos. 25 to 29.

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*This position is necessitated by the sharp curves around which nearly all mine cars must pass.

The mine car used at the Hollenback Colliery, shown by Page plate No. 25, is of comparatively small capacity ; but it will be observed that the guage is only two feet and seven inches, and for this guage the car is of more than average size. The bed frame consists of two main sills ten feet long, and six bed-sills 3′ $2\frac{1}{2}$″ long, braced by four iron tie rods (bolts) which also pass through and clamp the side post shoes. There is no center side post.

The draw-bars pass through all of the bed-frame sills, to which they are clamped by keys, and are connected together at the center of the car. The bumpers are very broad, but not deep.

The door lock is a rod provided with lips or catches at *b* and *b*. One end of the rod is turned up and held in position by a ring or link at "*a*," which is knocked up as the car runs on the dump.

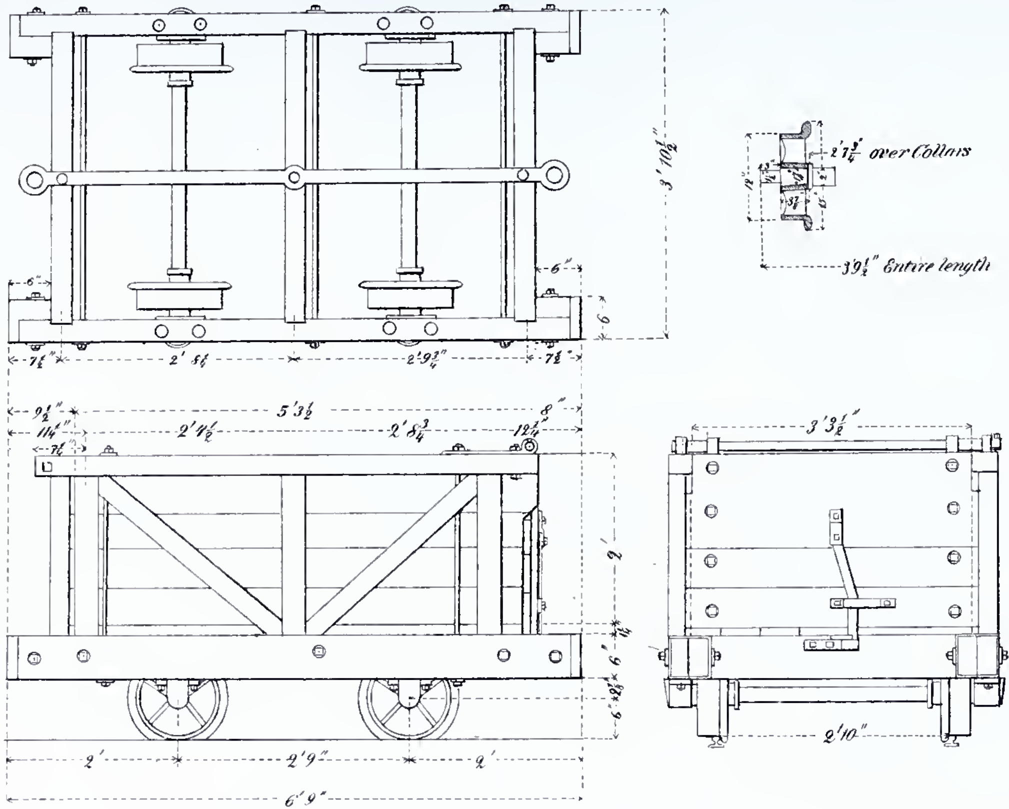
An enlarged section and front view of the oil box and seat are shown in the upper right-hand corner of the plate.

The extreme height of this car to the top of the door hinge is about four feet, its length ten feet, and extreme width (length of hinge bolt) six feet. Its inside dimensions are, length eight feet ten inches, height two feet, width at bottom three feet ten inches, and at the top four feet and ten inches. Capacity, when loaded even, about seventy-five cubic feet, loaded heaping full, ninety-five to one hundred and ten cubic feet.

Wheel base 3′ 6″; wheels 18″ in diameter; tread 3″; axles $2\frac{3}{4}$″ in diameter; journals $2\frac{1}{4}$″×5″.

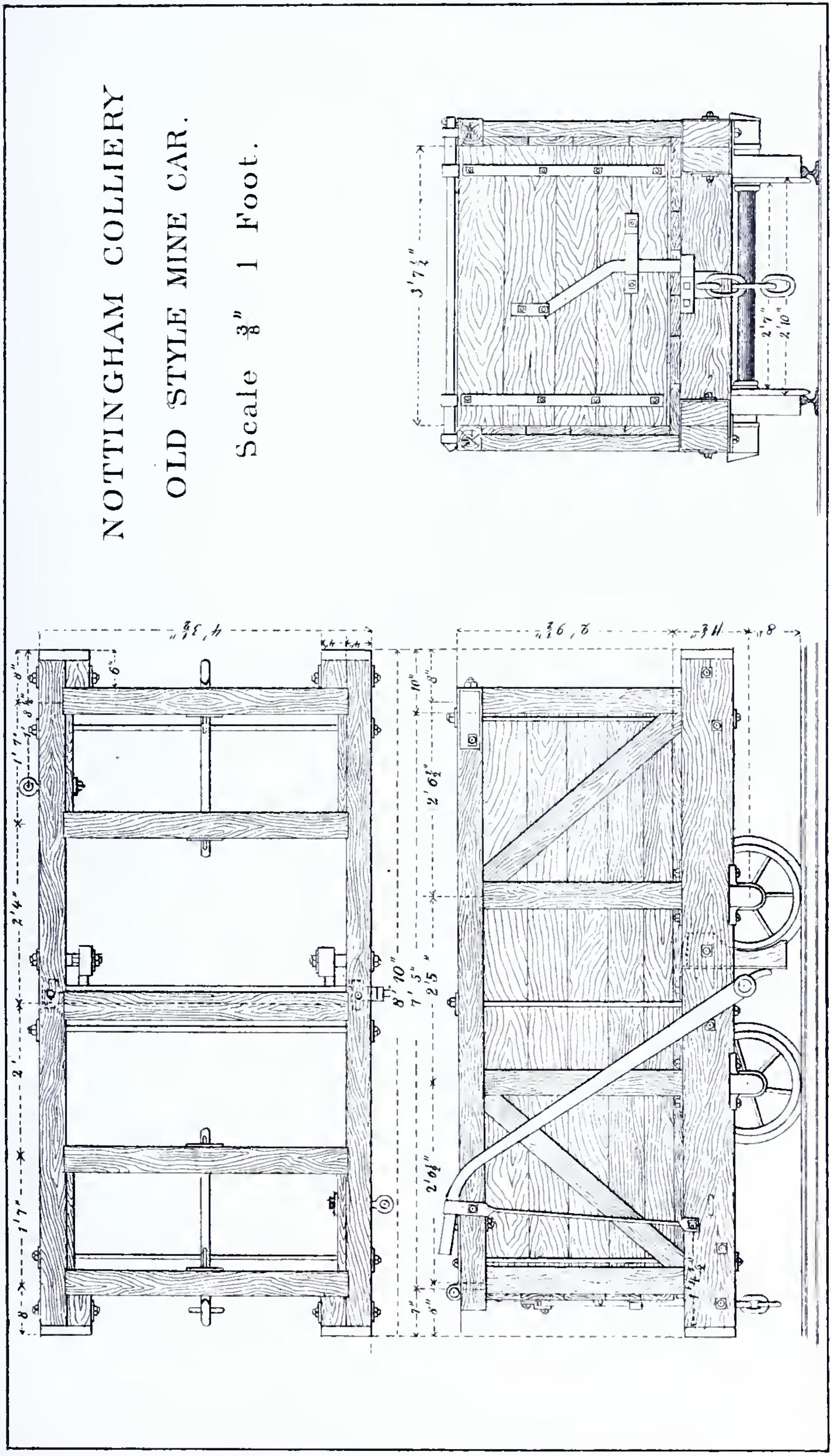
The style of car shown by Page Plate No. 26 is in more general use. It is a square car built for one of the broader guages (3′ 6″), and will hold, when loaded even to the guard, about one hundred and ten cubic feet. Its extreme length is ten feet and three inches, height four feet and ten inches, and width about five feet and three inches. The actual width necessary to pass the car is increased over six feet by the latches, which are made to project so that the

SMALL MINE CAR DESIGNED BY J. H. BOWDEN FOR BRODERICK & COS. MINES.



INSIDE DIMENSION

Length	5' 3½
Bradth	3' 3½
Depth	2 0.



NOTTINGHAM COLLIERY

OLD STYLE MINE CAR.

Scale $\frac{3}{8}$" 1 Foot.

A. Winslow Del.

car door may be automatically opened at the dump by the device shown by Fig. 59.

The arrangement of the bed-frame sills is not as well adapted to make a stiff frame as the one just described.

Peculiarities of this car which may be noted: Comparitively small width of bumpers; the odd form of draw-bar; bolts to stiffen the sides and hold the top rail; arrangement of bed-frame sills, side posts and braces.

It will also be observed that the width of the car is increased by standing the side posts in shoes or holders placed outside the main sill. The wheels with wrought iron spokes are also a noticeable feature.

A very small car—holding only about thirty-five cubic feet when loaded even—is shown by Page Plate No. 27. It is designed for a two foot ten inch guage, has a wheel base of two feet and nine inches, an extreme length of six feet and nine inches, a width of four feet, and is three feet and six inches high to the top of the hinge.

The simplicity of the bed-frame, which seems too light for strength, and the method of construction, show at once that it is a cheap car. Its small twelve-inch wheel will be noticed.

Cars of this size are now rarely used in the anthracite regions except at very small collieries.

Page Plate No. 28 shows a medium-sized old-style square car with brake. It is three feet seven and a half inches inside, seven feet five inches long, by two feet nine and a half inches high, holding therefore about seventy-five cubic feet.

It has already been stated that brakes are now seldom applied to mine cars, as the method of controlling the cars by spragging is much quicker and more effective.

The hanging latch and catch is similar to that shown by Page Plate No. 27.

The wheel base of this car is small (2′ 5″) for a car of this size and guage; the draw-hook is fastened in such a way that the bed-frame will soon be racked loose, the side posts are held by framing only, and the bumpers are narrow and rather thin.

It will be observed that the side-post braces are inclined from the center towards the bumpers; the present practice is to incline them in the opposite direction as shown on all the other plates.*

In the Eckley slope car shown by Page Plate No. 29 a staple replaces the ordinary form of draw-hook, and the strain is transmitted to the whole bed-frame through bolts connecting the sills together. This car is built with outside wheels, axles running in pedestals, for a four foot guage. It is nearly eleven feet long, five feet eight inches extreme height, and about five feet three inches wide; inside dimensions nine feet long, about two feet nine inches deep at front end and about three feet three inches at theback end, three feet eight inches wide at bottom; cubic contents when loaded even with top rail, therefore, about one hundred cubic feet or more.

Wheel base, three feet; wheels eighteen inches in diameter.

It will hardly be necessary to state that this car was especially designed for use at slope collieries opened on dips steep enough to allow the coal to run out over the top of an ordinary car.

One of the objects of placing the draw-hooks at the sides has already been indicated, but by the arrangement here shown the car can be run over a knuckle and dumped like a gunboat without being detached from the rope.

This is probably one of the best, if not the best slope car in use in the anthracite regions; it is substantially built, and the design, while simple, is one that should insure a good stiff car, needing few repairs, and such repairs should be cheaply made.

Weight of Cars.

The weight of mine cars in common use varies so widely with the size and method of construction that it is no more possible to state the average weight than it would be to state the average weight of railway cars.

Stiffly built, well-constructed cars usually weigh about

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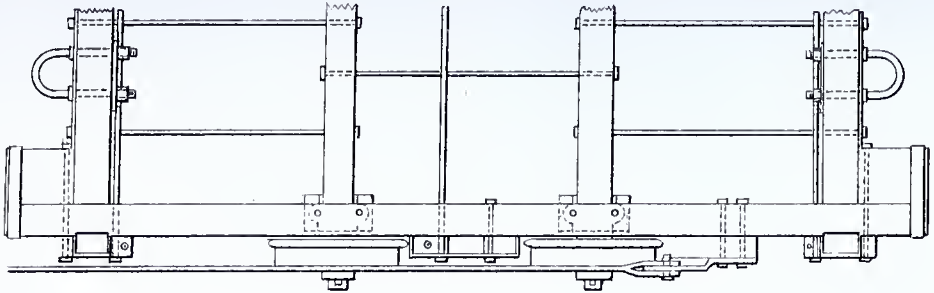
* I am not certain that the drawing from which this plate was made is correct in this particular.



ECKLEY SLOPE CAR

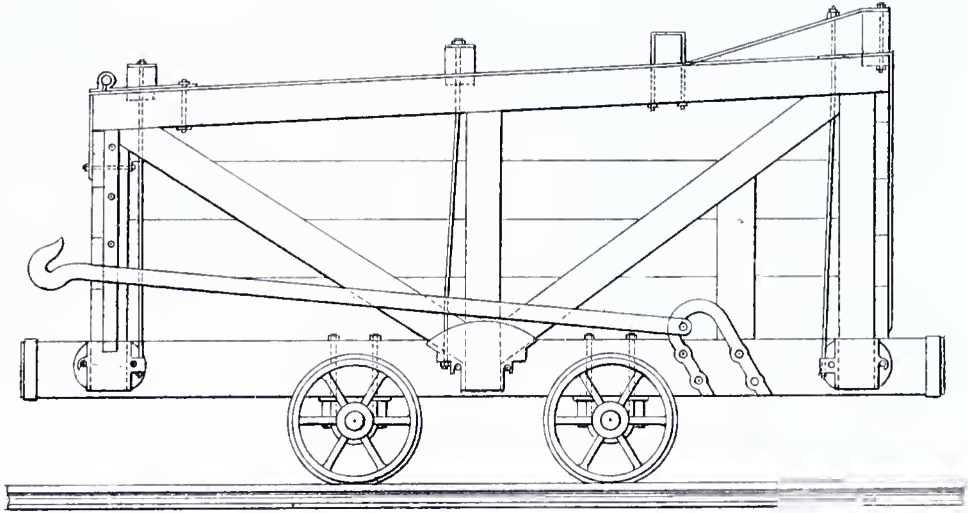
BUILT AT THE

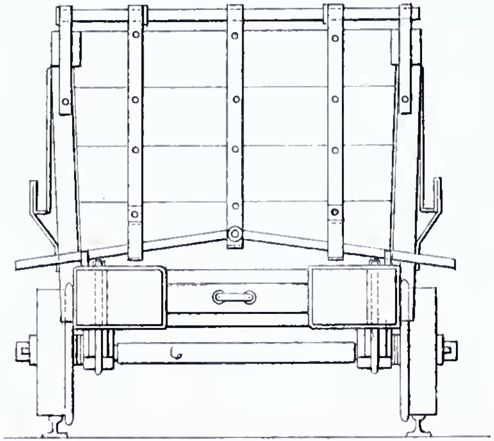
HAZLETON SHOPS

FOR

COXE BROS.

Scale $\frac{3}{8}$" — 1'.





one half as much as the weight of coal they are designed to carry. Their weight is therefore generally a few hundred pounds more or less than one ton, but some of the heavier cars weigh from one ton and a half to two tons.*

In considering general haulage and winding problems the weight of empty cars may be conveniently assumed at one ton each.

Underground haulage.

The motive power for underground haulage is supplied by mules † and locomotives.

Of the various mechanical systems of underground haulage in use in English, and to some extent, in American bituminous mines, only one—the tail-rope—has been tried in the Anthracite fields.

The plant alluded to is working at the Buck Mountain Coal company's colliery. I understand it consists of about half a mile of gangway track, leading from the foot of the slope to the breasts now working. The road contains several sharp curves. From fifteen to twenty-five cars can be taken out in each trip.‡

The famous report of the "Tail-Rope Committee," published in Vol. XVII (1867–8) of the Transactions of the North of England Institute of Mining Engineers, recognizes five distinct haulage systems as follows :

1. *Tail-rope system*, in which the full cars are taken out by the main rope ; the tail-rope being used to draw back the empties. This system is simply an amplification of the inclined plane operated by a barney with a tail-rope to draw the barney and main rope down the plane.

Branch gangway roads are provided with an independent rope and foot sheave, and when a trip of cars is to be taken into a branch, the main and tail-ropes are broken at shackles placed at the proper points on these ropes, and the branch road rope attached. Tail-rope haulage does not require a double road-bed.

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* On a preceding page the weight of two very heavy cars is given—both exceeding two tons.

†Horses are occasionally used, but mules are greatly preferred.

‡ Circumstances have unfortunately prevented me from visiting and examining this plant. If I am able to visit it before this report is printed, I shall describe it at greater length in the preface or prefatory letter.

2. *Endless Chain system.* The chain is carried *on top* of the cars which are strung along the road at intervals of ten, twenty, or thirty yards. The speed is very slow,—two to four miles per hour,—in strong contrast to the Tail-rope system, with its long train of cars running five or ten miles per hour. A double road is required.

3. *Endless Rope No. 1 system.* Rope beneath the cars; cars run in trips of several; attached to rope by hand clamp resembling a blacksmith's rod tongs. Cars sometimes attached by hook fastened in shackle on rope.

4. *Endless Rope No. 2 system.* Rope above cars; cars attached singly by small chain wrapped around the rope, or by hooking them to hemp loops lashed to the rope at intervals.

5. *South Wales Endless Chain system.* Chain beneath cars which are attached by hook and chain. Used on slopes and steep gangway roads. Double road necessary.

The Committee stated in conclusion that "as far as the cost of maintenance and working charges are concerned, the Endless Chain system can be applied, with few exceptions, to every condition of wagon-way with greater economy than any of the other systems; these exceptions being the cases:

"1st. Of the tail rope system where numerous branches are worked;

"2d. Of the No. 2 Endless Rope system, where the plane rises in one direction and has one or more sharp curves; and

"3d. Of the South Wales Endless Chain system, where the gradient is heavier than 1 in 12, or 1 in 8 where stations [*landings*] are worked. . . . .

"The No. 1 Endless Rope system is to be preferred to the tail-rope when the plane is undulating, has neither stations nor branches from the main way, and is with or without slight curves.

"The No. 2 Endless Rope system is only to be economically applied under the conditions stated above, and it cannot be advantageously adopted where the planes are undulating.

"The Tail-rope system is preferable to No. 1 and No. 2 Endless Rope system whenever any stations by the side of the plane,* or branches from the main way, (main haulage road,) are to be worked."

These conclusions were reached by the Committee after a most exhaustive investigation, and have been considered authoritative by English mining engineers; but whether they will prove applicable to Anthracite mining yet remains to be determined.

The cars in use on the roads examined by the Committee are very small as compared with the cars in use in this district. They varied in weight from 225 to 688 pounds, and carried only from 5½ hundred weight to half a ton of coal.

From the fact that the endless chain system requires a double road bed, one for the empty and one for the loaded cars, it seems inapplicable to anthracite underground haulage; as the cost of constructing and maintaining double track gangways for cars of the width now in use would be so great that it would probably exceed the entire cost of keeping open and operating an ordinary single track gangway.

The chief objection to the endless chain, when operated as described under § 5, is the loss of power by *friction;* but this only applies to long hauls. Chains of this description are frequently used on short inclined planes on the surface. (See Hammond Breaker Atlas Sheet, No. XVIII.)

The Endless Rope system No. 2 seems to be a "*make-shift*" arrangement, and would probably find few advocates among our mining engineers.

Endless Rope System No. 1 has more to recommend it, and as it can be used on a single track gangway, it may be adopted with advantage on straight roads without branches. When used on a single track gangway the return rope is carried on sheaves or pulleys on the side of the gangway or it may be placed beneath the road bed; the engine is reversed to draw in a trip of empties.

*** Used to mean road-bed.**

From the requirements of anthracite mining, and the conditions to which any haulage system must be adapted, it seems probable that the adoption of the tail-rope system will be found more feasible than any other.

The advantages which such a system might have over haulage by locomotive engines are:

1st. Absence of boilers and engines underground, the power being transmitted from the surface.*

And (consequently)

2d. Possibility of using system where locomotives are inadmissible,

3d. Decreased cost of haulage.

It is extremely doubtful whether the tail-rope or any other system of underground hauling will be found cheaper than haulage by locomotives, except on roads having *heavy grades*, or on *long hauls* of a *very large tonnage*.

The cost per ton per mile of hauling coal is given by the committee as follows:

SYSTEM.	Tons per day.	Distance in yards.	Average adverse grade.	Cost per ton per mile — pence.
Tail rope, ...	476	2133	1 in. 213	1.879
Endless chain, ...	451	1339	1 in. 59	1.379
Endless rope No. 1, ...	384	922	1 in. 48	2.061
Endless rope No. 2, ...	443	849	1 in. 36	2.993

Averaging the first three systems we find the cost 1.77*d.*, equivalent to say three and a half cents per ton per mile.

But the greater part of this cost is due to labor and materials that cost double the (English) price in this country,† which would make the cost here about seven cents per ton per mile. But from this figure a deduction must be made for the decreased cost of transporting the coal in so much larger cars and on more favorable grades.

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* In English mining practice the driving engine is commonly located underground.

† For example: 'Stationary engineer's wages, four shillings" (not quite one dollar) per day.

We may therefore conclude that the cost of haulage by any of these systems will be at least four to six cents per ton per mile, unless *the distance* the coal is hauled *is very great and the output large.* Haulage by mules costs no more than this on gangways with the favorable grades common throughout the anthracite region.

Rope haulage underground seems especially adapted to workings in comparatively flat coals where the gangways are often driven with heavy grades—in other words, to those collieries in which the mining conditions most nearly simulate those of the English and Welsh mines.

Rope haulage has been employed for several years—as far back as 1874 at least—at some of our bituminous mines in the vicinity of Pittsburg, and is now used at collieries in several other States.

It does not seem probable that it can be economically used in our anthracite mines to replace mule or locomotive haulage, unless there is more work than can be done by one locomotive, or by twelve or fifteen mules ; or at collieries in which, whether from the height of the bed, presence of gas, or scanty ventilation, locomotives cannot be used ; or where the grades are very heavy.

When the coal dips sharply and the bed is developed by the method of mining by lifts, rope haulage can only become valuable after the gangways have been driven a considerable distance from the outlet, and after mining has been stopped in the breasts near the outlet.

The double inside slope, operated by an "endless" rope, at Nanticoke No. 3 colliery, resembles in some respects both the No. 1 and No. 2 endless rope systems of the tail-rope committee's report.

This slope is shown in detail by the drawings of Atlas Sheet No. V.

The plant was designed for a true endless rope system in which the cars were to be hooked to clamps on the rope, but as the clamps cut the rope, and as they could not be held properly in place, they were replaced by introducing three chain links at the point at which the cars were attached to the rope, both before and behind. Five such places are provided (as shown by the detailed drawing in the corner of the plate) at each of which a trip of three cars* may be attached. There are, therefore, two rope sockets with three chain links in front of the trip, and two behind it, making four rope sockets for each of the five places at which trips are attached, or twenty in all.

The cars are so arranged that a trip of three loaded cars is starting up from the bottom, when a trip of loaded cars is starting *down* on the opposite side, and three empties are starting from the other side at the time that a trip of three empties is starting down from the summit.

By this arrangement the cars on one side nearly balance those on the opposite side and the power necessary to run the plane is reduced to a minimum.

It will be observed that the power is applied by two drums around which the rope makes two turns and then passes back to a sheave mounted on a tension truck. The details of this sheave-truck are shown by the upper right hand drawing.

The engine is geared direct to the nearest drum and the two drums are made to run together by a connecting rod.

The drums are eight feet in diameter, one foot wide, with four grooves. Back sheave, mounted on truck, eight feet in diameter, which equals the distance between center lines of the two tracks.

The foot-sheave is held in a frame placed in a pit beneath the level of the road-bed as shown by the illustration. This sheave is of the same size as the back sheave.

At first sight it would seem probable that the cost of operating a double plane of this description would be largely increased by rapid wear of the ropes. Twenty rope sockets—and each of these subjected to a more or less severe jerk from the cars every time a trip passes over the summit and begins its downward journey. Mr. Bowden states that they have not had a rope break on this plane for two years; that they handle over two hundred cars every day, and that the rope sockets give no trouble.

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* The drawing shows only two cars in a trip.

Without discussing the policy of building such a plane in preference to driving a tunnel, we are naturally led to inquire whether some other system of operating it would not give better economic results.

The Tail-rope Committee's conclusion that "where the gradient is heavier than 1 in 12," the "South Wales Endless-chain system" is preferable to any other, seems peculiarly applicable to this case.

Mule haulage.

Statistics contained in the Mine Inspectors' reports show that in 1880 there were about seven thousand mules worked in the anthracite regions. This includes mules worked above-ground as well as those employed in the mines.

Under the ordinary conditions of mine haulage, the detentions at turnouts, delays from accidents, imperfect condition of the road bed, the unevenness of the travelling way, and other peculiarities of mine haulage, reduce the daily average amount of work done by one mule far below the useful effect obtainable on surface roads of good construction,—probably to an average of not more than *twenty-five* to *fifty per cent.*

The average useful effect of one mule may therefore be considered to range from twenty to fifty tons hauled one mile per day. On roads of average grade we may assume the useful effect at forty tons.

If the car weighs one ton, and carries two tons of coal, the useful effect in *tons of coal* is twenty tons (of coal) hauled one mile per day by each mule. At some collieries the average is considerably larger, and at others smaller; but this* may be considered to fairly represent the daily performance of an ordinary mine mule, when averaged throughout the year.

Except when used for switching, mules are seldom driven singly. The team used to haul a trip of cars is commonly composed of two, three, four, five or more mules.

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* In arriving at these figures, allowance has been made for the time during which the animals are on the sick list. As accidents are continually happening to the mules, this is a factor of considerable importance. From five to fifteen *per cent.* of the mules used underground can usually be found in the stable, either disabled by accident, or on the sick list from overwork, or what not.

The height of the gangways usually permits the use of very large mules, and there are few collieries at which the small animals common in bituminous districts are preferred.

They are hitched together in a string, with spreaders between each two to keep the *tug-chains* apart ; the leader commonly carries an ordinary mine lamp fixed to a helmet attached to his head, or hung in a shield fastened to the collar.

It will hardly be necessary to refer here to the peculiar fitness of mules for mine work, nor to their long-proven superiority over horses for work of this class ; but it may not be amiss to refer to their proclivity for kicking, and the popular belief in their supposed stupidity.

Stupidity is one of the last charges that any colliery worker should make against his mules ; they are generally patient, easily handled, and wonderfully intelligent and quick to learn the peculiarities of their work. Here and there may be found a mule that has the reputation of being a "kicker," but the percentage of mules addicted to this vice is probably no larger than that of horses under similar conditions. The drivers or miners are themselves generally responsible for the kicks they receive.

After being used for a few weeks, an average mule will go through all the evolutions he must perform at the outlet and at the workings, with comparatively little trouble to the driver, and often without waiting for the customary orders.

Stabling is usually provided near the outlet. This location is chosen because feed and other supplies and the stable refuse do not have to be conveyed far, but principally because in case of accident or fire the probability of losing the mules is much less than if the stables were located far inside.

The stables generally open directly upon the gangway, or on a branch road, but are hedged off to prevent the air from sweeping too strongly through them ; they are generally well ventilated, but sufficiently warm, clean and dry—in other words, comfortable. In appearance inside, they resemble an ordinary stable above-ground, except that the roof and often nearly all the walls are of coal or rock.

In feeding, grooming, watering, bedding, and general attendance, mules stabled underground are as a rule as well cared for as those employed aboveground.

The number of cars taken at one trip by a team of a certain number of mules varies with the weight of the cars, their capacity, and the grade and condition of the road-bed.

On gangways of average grade (a favorable grade) the trains vary from one to three cars for each mule in the team. When the gangway grade is from one foot in one hundred to one foot one hundred and fifty feet (fall towards the outlet) the power required to draw the loaded cars out is nearly the same required to draw the same number of empty cars in, so that the number of cars is *not* limited to the *number of loaded cars* a team can draw.

But as heavy grades often occur, especially in flat workings, and as these grades are nearly always towards the outlet, the number of cars in a trip is frequently limited by the *number of empty cars* the team can draw back over an adverse grade.

Mine Locomotives.

It is apparent at first sight that mine locomotives can be economically employed only on comparatively long hauls.

When there is sufficient haulage exclusive of switching, handling single cars, etc., to keep eight or ten mules constantly at work, it is generally considered advisable to replace the mules by a locomotive.

Locomotives of the size now commonly used, weighing from seven to nine tons when loaded for service, are variously considered to be equivalent in effective work, to from ten to twenty large mules. This wide variance in the estimated effective duty is due not so much to the differences in the engines, as in the conditions of the road-bed and mine ventilation, and in the character of the work performed.

When the engine is used largely for switching and for handling empty cars the work performed does not compare as favorably with mule haulage as when the locomotive is used on long hauls; the presence of sharp curves and an imperfect road bed, and the presence of doors on the gangway, limit the speed at which the trains may safely run, and thus reduce the useful effect far below what can be realized from the same engine when worked under more favorable conditions.

The speed at which trains are run rarely exceeds seven miles per hour, and as a rule averages not quite five miles per hour, while with mule haulage the average speed on ordinary grades is from two to three miles per hour, or less.

The locomotives in use are all coal burning, and the gases, smoke, and exhaust steam escape direct into the gangway. It has frequently been proposed to replace the boiler by an air receiver, to be charged with air from a compressor located on the surface; but while this plan would probably be perfectly feasible, its cost has as yet prevented its adoption.

Those who have walked through a long railway tunnel after the passage of two or three locomotives can readily picture the condition of a mine gangway and workings in which a locomotive is used, and in which the ventilation is rather sluggish.

The use of mine locomotives is restricted to well ventilated, non-gaseous gangways, and to mines in which no trouble is experienced from the accumulation of carbonic acid gas, (*choke-damp*, *black-damp*, CO_2.)

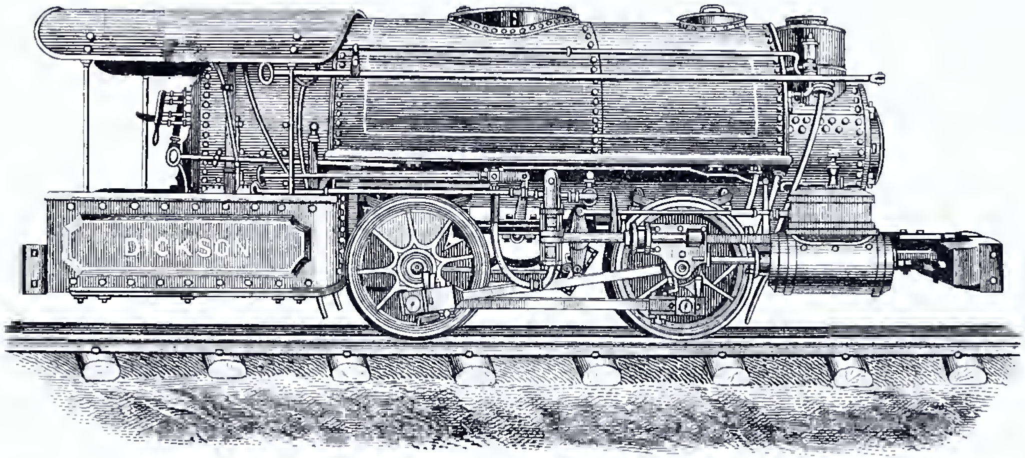
In addition to the objection that mine engines heat and foul the ventilating current, another objection, equally if not more important, is frequently advanced by those opposed to their use, viz: The increased risk of fire. It will not be necessary to indicate the multitude of ways by which a mine fire may originate from the locomotive boiler fire.

It is evident that this risk is much greater than from stationary fires or machinery underground, and it is equally evident that the danger will, as a rule, be dependent mainly upon the watchfulness of those in charge of the locomotive engines. Whether it is a wise policy to thus place the safety of the whole mine in the hands of one or two men

MINE LOCOMOTIVE USED BY THE D. & H. COAL CO.

DICKSON MANUFACTURING COMPANY

Scranton, Pa.



Weight 7½ Tons
Guage 3 Feet

Height 5'2"
Wheel Base 4'3'

Cylinder 8 × 12 Inches.

who may or may not prove reliable, is a question that must be decided by each superintendent and engineer.

Under the careful management of reliable engineers, the use of locomotives underground should increase the (already and always present) risk of fire very slightly.

Before accepting this additional risk, it would seem good policy to inquire minutely into the annual saving that may be effected in any particular case by introducing locomotive haulage, to determine whether this is sufficient to warrant the acceptance of this risk of fire, the presence of waste gases, etc., etc.

It is claimed that on hauls of moderate length a saving of from one to three cents per ton per mile may be effected by substituting locomotives for mules.

I have not been able to obtain any exact figures for comparison of these two systems of haulage, because it is extremely difficult to estimate the increased cost of maintenance of road-bed and the amount that should be charged against the locomotive for increased wear and tear on rails, sills, frogs, crossings and switches.

The accompanying table is reproduced here from the Inspectors of Mines Report for 1880, (See pages 222 and 223).

Inspector T. D. Jones makes the following comments on the data contained in this table:

" Where there are two locomotives used, one inside and the other outside, only that of the inside one is taken into account. Hence, the averages have been obtained by dividing by nineteen. It will be observed that the average difference, in first cost, of mules and locomotive is $852 90, and the mean difference in favor of the latter is $5 97 per day.

"To use the mine locomotive for underground haulage the following conditions should be taken into consideration: (1.) Adequate ventilation. (2.) Ventilation produced by mechanical appliances, that by fan preferable. (3.) Velocity of the air current should be from eight to twelve feet per second, and not less than six feet. The mean speed of the locomotive is about seven feet a second, which is a trifle less than the former velocity of the air current advo-

Table showing the Comparative Work and Cost of Haulage by Locomotives and Mules.

COLLIERIES.	BY WHOM OPERATED.	NUMBER OF LOCOMOTIVES.		DIMENSIONS OF LOCOMOTIVES.						Cars hauled per trip.	Weight of load—tons.	Distance traveled in round trip—miles.	Time taken to run round trip—miles.
		Inside.	Outside.	Steam pressure per sq. inch—lbs.	Stroke—inches.	Cylinder—inches.	Drivingwheels—inches.	Weight—tons.	Estimated horse power.				
1. Ebervale, No. 2,	Ebervale Coal Co.,	1		130	14	9½	27½	9	27	10	37½	0.530	6
2. Room Run, No. 3,	L. C. & Nav. Co.,	1		110	16	9	27	10	32	12	48	1.708	18
3. Room Run, No. 2,	L. C. & Nav. Co.,	1		110	10	6½	30	4	16	10	40	1.250	32
4. Panther Creek, No. 9,	L. C. & Nav. Co.,	1	1	110	12	9	30	8	32	25	100½	0.833	13
5. Panther Creek, No. 6,	L. C. & Nav. Co.,	1		85	12	7	28	5	18	12	48	1.478	24
6. Panther Creek, No. 5,	L. C. & Nav. Co.,	1		100	12	9	30	8	32	25	100½	2.242	28
7. Oak Dale, No. 1,	G. B. Markle & Co.,	1		120	14	9	28	7	24	14	55	4.091	50
8. Oak Dale, No. 1,	G. B. Markle & Co.,	1		90	14	9½	28	9	27	10	34	0.544	10
9. Laurel Hill,	A. Pardee & Co.,	1		100	14	8	28	5¾	18	13	50	2.272	23
10. Lattimer, No. 2,	Pardee Bros. & Co.,	1		120	14	9½	30	8	25	3	50	1.307	13
11. Mount Pleasant,	Pardee & Sons,		1	100	14	9½	28½	7	20	10	30	1.137	20
12. Upper Lehigh,	Upper Lehigh Coal Co.,		1	90	14	9½	30	8	25	10	32¼	1.328	15
13. Cross Creek, No. 1,	Cox Bros. & Co.,	1		120	14	10	28	7	25	7	33	1.054	20
14. Cross Creek, No. 2,	Cox Bros. & Co.,	1		110	14	10	28	7	25	8	38	1.440	20
15. Coleraine,	W. T. Carter & Co.,		1	110	14	9	30	9	30	6	22	1.477	15
16. Tresckow, No. 6,	E. B. Leisenring,		1	100	12	8	30	7	30	10	35	0.981	10
17. Stout, (Milnesville,)	Stout Coal Co.,		1	120	10	8	30	7	30	12	36	1.136	10
18. Beaver Brook,	C. M. Dodson & Co.,		1	80	12	7	20	5	20	5	20	1.710	20
19. Humboldt,	Lindermen, Skeer & Co.,		1	65	13	9½	28½	8	32	10	31	1.510	8
Averages,		12	8					7.3	25.68	11.63	44.25	1.475	18.68

Table showing the Comparative Work and Cost, &c.—Continued.

COLLIERIES.	Rate of speed in miles per hour.	Grade—degrees.	COST OF RUNNING LOCOMOTIVE PER DAY.				COST OF HAULAGE BY MULES PER DAY.				Difference in cost per day, in favor of mine locomotives.	VALUATION OF	
			Engineer and fireman.	Packing, oil, coal, etc.	Repairs.	Total.	Number to do the same amount of work in the same time.	Feed, harness, shoeing, attendance, &c.	Drivers.	Total.		Locomotive.	Mules.
1. Ebervale, No. 2,	5.300	1 to 3	$4 25	$0 50	$0 60	$5 35	15	$9 00	$7 00	$16 00	$10 65	$3,000	$2,625
2. Room Run, No. 3,	5.693	1 to 4	3 48	50	25	4 23	12	4 80	3 01	7 81	3 57	4,000	1,920
3. Room Run, No. 2,	2.344	1 to 3	3 48	48	35	4 31	6	2 40	3 07	5 47	1 16	3,000	960
4. Panther Creek, No. 9,	3.845	1 to 2	3 65	82	25	4 72	12	3 24	4 00	7 24	2 52	3,000	1,920
5. Panther Creek, No. 6,	3.695	1 to 3½	3 65	82	30	4 77	12	3 24	4 00	7 24	2 47	2,500	1,920
6. Panther Creek, No. 5,	4.804	1 to 3½	3 65	82	1 60	6 07	18	4 86	6 00	10 86	4 79	3,000	2,880
7. Oak Dale, No. 1,	4.908	1 to 3	3 50	75	26	4 51	18	5 40	5 25	10 65	6 14	2,700	3,200
8. Oak Dale, No. 1,	3.264	1 to 3	3 50	75	26	4 51	10	3 00	2 00	6 00	1 49	3,000	2,000
9. Laurel Hill,	5.927	1 to 2	2 72	86	1 76	5 34	16	4 96	6 84	11 84	6 50	3,800	3,200
10. Lattimer, No. 2,	6.033	1 to 3	2 58	1 50	1 00	5 08	25	11 15	8 35	19 50	14 42	2,900	5,000
11. Mount Pleasant,	3.411	1 to 2	1 98	1 06		3 04	8	2 48	2 40	4 88	1 84	3,200	1,600
12. Upper Lehigh,	5.312	1 to 3½	1 80	1 00	05	2 85	16	4 80	2 30	7 10	4 25	3,500	1,000
13. Cross Creek, No. 1,	3.162	*1 to 4	3 94	2 00	1 70	7 64	16	16 00	12 88	28 88	21 24	3,000	2,880
14. Cross Creek, No. 2,	4.320	1 to 5	2 91	2 00	62	5 53	12	12 00	9 65	21 65	16 12	3,000	2,190
15. Coleraine,	5.908	1 to 6	3 00	75	15	3 90	8	3 04	5 00	8 04	4 14	3,000	1,000
16. Tresckow, No. 6,	5.886	1 to 5	2 75	50	12	3 75	6	2 10	5 00	7 10	3 35	3,500	1,200
17. Stout, (Milnesville,)	6.816	1 to 5	2 75	25	10	3 10	6	1 75	2 40	4 15	1 05	2,300	900
18. Beaver Brook,	5.630	*1 to 4	2 55	1 25	25	4 05	8	3 20	3 00	5 20	1 15	2,000	1,800
19. Humboldt,	‡11.325	1 to 3	2 61	75	25	3 61	16	4 50	5 60	10 10	6 49	2,800	2,800
Averages,	5.136	1 to 3½	$3 09	$0 92	$0 52	$4 54	12	$5 36	$5 21	$10 51	$5 97	3,010.53	2,157.63

* Engine hauls loaded trip up grade. † Only an engineer required.

‡ This should be thrown out in estimating the *average speed*, which is about 4.8 instead of 5.136.—H. M. C.

cated. (4.) The size of gangways and tunnels, where locomotive travels, should not be less than seven feet high by ten feet wide; although there are some running, at a few collieries, in places not more than seven feet by eight feet. "Of course, the more room the better. (5.) The locomotive track should be kept in good condition, which is not a very easy matter to do in mines that are very wet, as the water from the shutes and ditch gets under the railroad ties, causing the joints of the T rails to become sagged. (6.) The engine run should be from tunnel mouth, bottom of shaft, or foot of slope, as the case might be, to inside tunnel or siding; at any rate, men ought not to be permitted to work on the route the locomotive travels, ..."

"The rest of the data are reliable, as they were obtained from the company's officials."

"Accidents to mine locomotive hands, in this district, are comparatively few. The only case resulting in death that has occurred was an engineer at Ebervale colliery, who was fatally injured by a car jumping the track, knocking out a couple of sets of gangway timbers, which fell on the engineer when reversing his engine, while those to mule drivers have been six this year. Of course there is a limit to the use of the mine locomotive for underground haulage, and I cannot advocate or permit its use further than that of hauling the coal from the inside turnout to the bottom of the slope, or along a portion of the gangway where the breasts are finished."

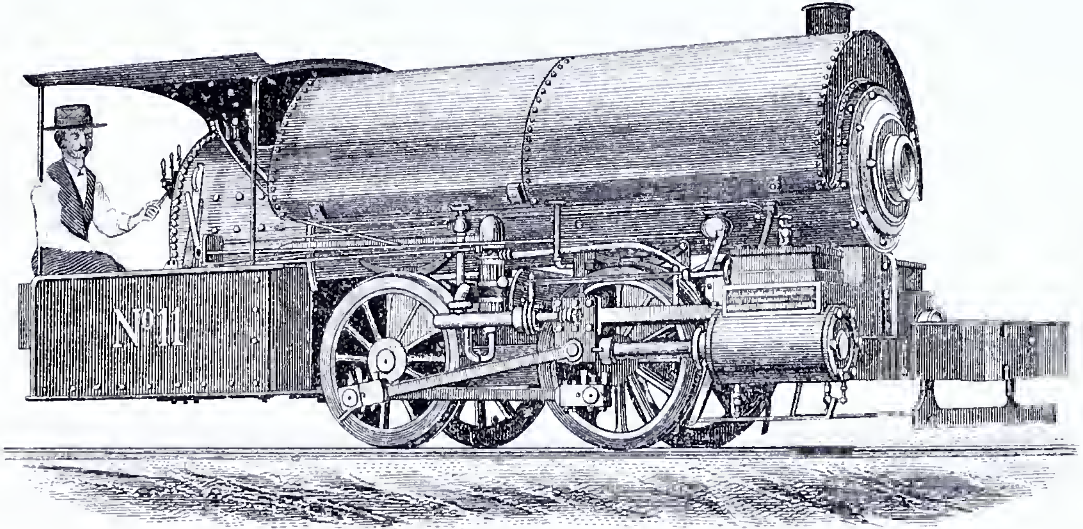
"Wherever the locomotive is used in this district the inlet for the air current is inside of the terminus of the locomotive run."

The data contained in the above table show that not quite nineteen minutes is required for a round trip of about one and a half miles, which would enable the engine to make at least twenty trips per day, hauling at each trip forty-four tons—a total effective duty of eight hundred and eighty tons hauled three quarters of a mile per day, or six hundred and sixty tons one mile per day.

Deducting the weight of the cars (about one third) we have four hundred and forty tons one mile per day. In actual practice, however, the engines do not average

MINE LOCOMOTIVE BUILT BY THE WYOMING VALLEY COMPANY

WILKES-BARRE, PA.



Diameter of Cylinders 8"
Stroke 14

Weight 8 to 9 Ton

much more than one half the above number of trips per day, so that the actual duty is probable from two hundred to two hundred and fifty tons of coal hauled one mile per day at a cost of four dollars and a half, the *cost per ton per mile* then being *about two cents*, and the cost of mule haulage from *four to five cents, or less.*

This table seems to contain some very conflicting data; thus the cost of feeding, shoeing, and attendance as given, ranges from 27 cents to $1 00 per day per mule; the cost for driving from 14 cents to 80 cents per day per mule, etc., etc.

An investigation by Messrs. Powell and Risdale, published in the School of Mines Quarterly, shows that at Drifton No. 1 colliery the cost of haulage by locomotive per ton per mile was 1.3 cents, and by mules 4.4 cents. At No. 2 colliery the cost was respectively one and three cents, the grade being more favorable.

The mine locomotive shown by Page plate No. 30 is manufactured by the Dickson Manufacturing company of Scranton. It weighs about seven tons when loaded for service, is three-foot gauge, and four feet three-inch wheel base. The cylinders are horizontal, eight inches in diameter by twelve-inch stroke. Driving wheels twenty-six and a half inches in diameter, with steel tires one and three quarters inches thick. Cast centers twenty-three inches. Driving axles of hammered iron four inches in diameter.

Tubular boiler of charcoal iron, with fifty-six tubes and steel fire-box; no combustion chamber. Grate area somewhat less than five feet. Dome eighteen inches high, with safety valve, whistle, and balanced throttle.

The tank is mounted on top of the boiler and has a capacity of two hundred and twenty gallons.

This engine is rated for service, on a surface railroad, as follows:

"On a level,	375	gross tons.
20-foot grade,	190	" "
40 " "	125	" "
60 " "	95	" "
80 " "	75	" "
100 " "	60	" ""

Its performance underground will, of course, be governed by the condition of the road-bed. On ordinary mine roads the tonnage is not much more than half the weight that can be hauled on a surface road in average condition.

On a tolerably straight gangway road of average grade it will not be safe to estimate the average maximum efficiency at more than sixty tons.

The engine shown by Page plate No. 31 is manufactured by the Wyoming Valley Manufacturing company of Wilkes-Barre, and is in use at several collieries near that place. Its cylinders are eight by fourteen inches, horizontal, and outside connected. Weight in working order, eight to nine tons.

Many other forms of mine locomotives have been built, and some of the older styles, of which a few still remain in use, are of unique construction. The Lehigh Coal and Navigation Company have used an engine in which the cylinders were set a steep angle of inclination in the body of the locomotive, and the power transmitted to the driving axles by cog gearing. Vertical boilers have also been used ; and inside connected engines have not been uncommon.

All of these types are gradually being replaced by engines belonging to the same class with those illustrated by Plates 30 and 31. These engines have no tender ; the coal is carried in a coal-box inside the cab.

The height and width of mine locomotives are, of course, governed by the size of the gangways on which they travel. The moving parts are, if possible, so placed that the water tank projects beyond them, thus affording protection from coal or rock that may fall from the gangway roof or from shutes or manways opening on the gangway. The wheel-base is, of course, governed by the gauge and curves around which the engine is to run. Mine engines as built at present are usually from five and a half to a little over six feet high.

CHAPTER XIV.

Winding Engines and Drums.

The engines in use for winding coal in the anthracite regions are usually built with horizontal cylinders and ordinary slide valves.

Steam engines of other types, the rotary, vertical, and oscillating, have been occasionally used, but the horizontal engines outnumber these ten to one.

A very large number of geared engines are in use, which are known as second and third motion engines, in contradistinction to the direct acting or first motion engines, but since the introduction of powerful double direct-acting engines, these are gradually disappearing, and at present the latter type is almost exclusively adopted at large newly opened collieries.

A general discussion of the steam engine is foreign to the objects of this report, but it is in place here to consider in detail some of the features peculiar to well-designed winding engines for colliery duty.

Since the opening, in the last few years by shaft collieries, of large areas of bituminous coal in Ohio, Indiana, Illinois, and other western States, a large number of western manufacturing firms have begun the manufacture of winding engines. These engines which are, many of them, extremely well-built steam engines seem to have been designed without reference to the fact that any good stationary engine geared to a drum may not make a good winding engine. In fact, the majority of such engines are unfit for colliery work, and at present there are very few machine shops located at a distance from the anthracite fields turning out winding engines suitable for this class of work.

It will be understood that these statements are not intended to refer to the mechanical execution of the work, but to neglect of certain details which long and dearly bought experience in the manufacture and repair of winding engines has taught the mechanical engineers of the anthracite region,—an experience as yet obtained at few localities in western coal-mining States.

The principal requirements of anthracite winding engines may be briefly stated :

1st. The engine must be thoroughly under the control of the engineer, so that it may

a, Be quickly stopped when running at full speed, and

b, Be moved with certainty and nicety through a small fraction of a revolution. This is necessary in landing.

2d. It must be capable of being quickly started with full load at any part of the stroke.

3d. Must be capable of attaining full speed in two or three revolutions, (with average working load.)

4th. Great strength of every part is absolutely essential to prevent frequent breakage from the severe shocks to which winding engines are always subjected.

5th. The last object is best attained by *simplicity*.

6th. To facilitate repairs, every part of the engine and drum should be easily and quickly accessible.

7th. Economy in the use of steam is often the last feature considered. As the coal burnt under the boilers is usually fine coal of comparatively little value at the colliery, *economy of fuel* is often of not much importance, but when the water supply is very impure and often insufficient, (as is frequently the case,) economy in the use of steam is most desirable.

These being the requisites of a good winding engine, we may consider the details of construction by which they can best be fulfilled.

Valves and Valve motion.—Few mining engineers feel favorably disposed towards engines in which the steam is used expansively. Under conditions most favorable to the use of engines of this class, the results attained cannot be considered satisfactory.

The valve gear cannot be arranged to act permanently for any considerable degree of expansion, because the engine may be required to start with full load from any point of the stroke at either the top or bottom landing, at intermediate landings, or at any point in the shaft during examinations or repairs.

Devices for bringing an independent cut-off automatically into action at a certain point in the run, are objectionable because they increase the cost and *destroy the simplicity* of the engine; the use of a cut-off that must be hooked up by the engineer is out of the question.

These objections to the use of an expansion valve-gear disappear when the engine is used to wind by friction gear, (as at the Cross Creek No. 2 colliery,) as the engine then runs continuously in one direction,—the speed is regulated by a governor.

The slide valve is used to the almost entire exclusion of other forms; the ports are large, and wire-drawing is especially guarded against.

The eccentrics are usually placed on the main driving shaft, which (in first motion engines being the drum shaft) is sometimes so large that the eccentrics are unwieldy and heavy, causing a considerable loss of power from friction, besides being in the way of repairs, and more or less difficult of access.

In the Pottsville shaft plant the eccentrics are carried on an independent shaft, taking motion from a reverse drag crank. Spur gearing to drive eccentric shafts is open to objection because the play of the teeth communicates a constant and injurious vibration to the valves; the noise they make is also a more or less serious objection.

Cornish valves have not been adopted,—with one exception,—but in the future we shall probably see many large winding engines built with these valves. It will not be necessary to enumerate the reasons why the slide valve is not well adapted to large engines. On engines of moderate size the slide valve is undoubtedly the best form that can be adopted; but for very large winding engines the cornish valve is undoubtedly superior to the slide valve.

But the slide valve is simplicity itself, is cheap, and there is but one valve chest to open when an examination is necessary,—merits that have and will, cause it to be preferred by most mining engineers.

The piston-rods are made stronger than in ordinary steam engines. This is necessary to enable them to withstand the shock caused by throwing the steam against the piston when the engine is running at high speed. They usually measure from one sixth to one seventh of the diameter of the steam cylinder.

Exactly the same necessity exists for making the connecting rods of corresponding strength.

In the anthracite regions back piston-rods are almost never seen; the largest engines are built without them.

Theoretically it may seem that engines of large size, at least, in which the pistons and piston-rods are very heavy, should be provided with back piston-rods; but the loss in effective piston area, the loss from friction in the back stuffing-box, and the increased cost, have been shown by practical experience to more than counterbalance (in winding engines) any advantages secured by adopting them.

Journals.—In direct-acting engines the main driving shaft,—crank shaft,—carries the drum, and the journals have, therefore, to sustain a weight of from fifteen to thirty tons or more. This weight includes the weight of the drum, about half the total weight of the rope, the weight of the shaft, cranks, connecting-rods, and eccentrics.

To properly distribute this pressure, and to sustain the shocks to which the crank shaft is subjected, the journals are given a long bearing. A rule adopted with large engines is to make their length equal to about twice the diameter; on small engines the length is often relatively greater.

Drum shafts.—Crank shafts of sufficient diameter to give the requisite margin of safety over the tortional strain to which they are subjected, will generally be large enough to give an ample margin of safety over the transverse strain produced by the weight of the drum and pull of the ropes. In other words, the thickness of the crank shaft is usually governed by the tortional strain to which it will be subjected. The factor of safety is large ; never less than 6, and often 10 or 12.

Pedestals.—The prevailing practice is to seat the journals in ordinary upright pedestals, but some large winding engines are built with angle pedestals. Whether this is necessary, or even advisable, is an open question, but the practice of making the inclination of the pedestal either parallel to or at right angles to the average inclination of the ropes is undoubtedly wrong in principle.

In adopting angle pedestals the guiding principle is to make the angle such that the center line of the pedestal shall be parallel to (or at right angles with) the *resultant* of all the forces acting on the journals. Neglecting the force exerted by the engine—and this may be neglected because it acts radially with equal power in opposite directions—the weight of the drum, shaft, eccentrics, etc., and the pull of the ropes are the only forces that need be considered.

As the verticle force, the weight of the drum, rope, crank shaft, etc., is usually two to four times as great as the pull of the ropes, the *resultant* is a force only slightly inclined from the perpendicular, and the angle to be adopted for the pedestal is so slight that there seems to be no reason why the ordinary straight pedestal should not give perfectly satisfactory results.

This is the view held by a large number of engineers and commonly carried out in practice ; but some think the inclined pedestal with the cap at right angles to the line of pull (inclination parallel to rope) should be preferred for large engines.

The Laurel Hill engine, Atlas Plate No. IV, is built with angle pedestals.

Broken pedestals (especially the cap) are of the most common occurrence. It is extremely difficult to make a pedestal that will stand the shocks and strains without making it entirely disproportionate in size to the engine and crank shaft.

Cranks.—Wrought-iron cranks are preferred for large engines. The difficulty with cast-iron cranks is that if they are put on too tight they break, if not tight enough they soon work loose.

The cranks of the Pottsville engine are models of what large cranks should be. On engines of moderate size both wrought and cast cranks are in common use, but the preference of engineers seems to be surely turning towards the exclusive use of wrought iron for both large and small engines.

The horizontal engine, with the drum directly in front of the engineer, a form that may be considered almost typical of an anthracite mining plant, is open to the objection that the height of the drum usually prevents the engineer from seeing the shaft mouth.

Whether this is an objection, whether it is not better to isolate the engineer and make him absolutely dependent upon signals, even in an emergency, whether he should be allowed to see the shaft mouth even if the plant can be so arranged, whether his attention may not be diverted from his duty, or his coolness forsake him at sight of imminent danger or actual accident at the shaft mouth, are questions to which engineers give varying replies.

The most common arrangement is that shown by Atlas plate No. IV, of the Laurel Hill engine, and Page plate No. 33, in which the engineer is placed directly in front of the drum in full sight of the links, the brake, the indicator, and the rope, with the reversing lever at his right and the throttle *overhead*, the lever hanging down so that the handle is about shoulder-high.

Foundations.

Engine foundations are, almost without exception, built of stone. Finish is not considered of so much importance as size sufficient to insure absolute stability.

No trouble is ordinarily experienced in obtaining good ground on which to seat the foundation pillars, the soil generally being sufficiently firm, but is often so shallow that the masonry is built on bed-rock.

Any sandstone that may be found in the neighborhood suitable for the purpose is used; loose rocks lying on the surface are often split up and utilized. The stone is rarely dressed with care, as it is cheaper to build large pillars of roughly-dressed stone than to dress the stone carefully and make the foundations somewhat smaller.

In height the foundation pillars are governed by the amount of clearance needed for the drums or any connections located beneath the engine floor—in other words, the cellar room necessary—and by the amount of stone necessary to insure absolute stability.

They are, therefore, generally made larger than those built for ordinary stationary engines.

The practice is to build the holding-down bolts in the masonry; no bolt holes are left. While this plan is open to several objections, it is undoubtedly superior in some respects to any other plan, and has been found by experience to be the best and cheapest plan. With accurately constructed templets and proper supervision of the work, no trouble need be experienced in fitting the bed-plate on the bolts.

One of the chief disadvantages of this method is that it does not allow the engineer to level the top of the foundation and bring the bed-plate to a perfectly true bearing by moving it to and fro. This objection does not hold good when the proper bearing is obtained by the use of a cementing mixture.

The plan adopted at the Pottsville shaft was very different from the ordinary method. The foundation pillars were completed without either inserting the bolts or leaving bolt holes, and these were then drilled with a diamond drill. I can see no reason why this may not be recognized as a most excellent method, as it seems to combine the advantages of the plan in common use with those obtaining to foundations in which bolt holes are left open.

While the main body of the foundation pillars is usually built of roughly-dressed stone, more attention is given to the upper courses, and the stone for these is carefully dressed.

It is barely necessary to refer to the importance of properly seating the bed-plate upon the foundation pillars, so that it shall have a perfect bearing throughout.

The use of timber for engine seats is now limited to small sinking engines, engines erected for temporary use, and to small engines in which the whole bed-plate is cast in one piece.

For winding engines, timbered seats are not considered allowable. It is true that some winding engines now in use are mounted on timber seats, but they are nearly all old second and third-motion engines built in the past, when timber seats were not so generally condemned.*

Mention has been made in a preceding chapter of the frequent use of timber crib foundations, and of the use of heavy sills for inside-slope engine foundations. As engines of this class are usually second or third-motion geared engines of moderate size, the necessity for an absolutely rigid foundation is not so great, and a good degree of stability may be obtained by notching and wedging the sills into the rib on each side of the engine-room or fitting them into the floor.

The adoption of masonry pillars for engines located underground is now becoming a common, and by many engineers is considered the only good, practice. When the bed is thin the cost is greatly increased, because a considerable quantity of rock must be taken up from the floor; or the roof rock must be shot down, to give sufficient head-room. When the bed-rock (floor) is a hard solid stratum, the bolts may be secured in holes drilled at the proper points in the floor, and the pillars made high enough to give clearance for the drum and working parts.

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* In the bituminous mining districts of Ohio, Indiana, and Illinois, timber seats are largely used. It is claimed that the timber seat furnishes an elastic medium to deaden and distribute the shocks to which the winding engine is subjected. That this is true will not be denied, and such a seat may often prevent breakage in an engine not properly constructed to bear these strains; but the timber seat keeps the whole engine in a *constant state of vibration*,—a condition fatal to long service in any engine,—its expansion or contraction may shift the engine from its proper position, dry-rot may cause serious accident, or, at least, the annoyance and expense of reseating the engine. *Rigidity*,—the great *desideratum*,—cannot be obtained with a timber seat.

The winding engine shown by Atlas plate No. VI, was chosen as a typical illustration of a direct acting, or first-motion hoisting engine, because it shows the application of two auxilliary appliances, not generally in use, viz: the steam brake and the steam reverse.

The cylinders are thirty-two inches in diameter, by six feet stroke. It will be observed that the throttle-valve lever *b*, the reversing lever *c*, and the lever by which the steam break is controlled, are situated within easy reach of the engineer when stationed on the left hand side of *c*. He then can handle the reverse *c* with his right hand, the throttle lever *b* (overhead) with his left hand, and the brake-lever is directly in front, within reach of either hand.

The construction of the drum, which is sixteen feet in diameter, is plainly shown by the drawing. The centers are of cast-iron,, strengthened by twelve ribs, bounded by a circular rib, outside of which are twenty-four small ribs, forming twelve sockets for the arms. The arms of wrought-iron, 8″×1″, bolted with three bolts to the center-piece, the lagging of eight-inch stuff, bolted by bolts with counter-sunk heads to the cast-iron angle-rims, which are made in six segments. The brake is a heavy band of wrought-iron, eight inches wide, by one half inch thick.

This engine is one of the largest and most powerful in use in the anthracite region.

Steam brake.

I have seen but three steam brakes in successful use in the anthracite region, viz:

Laurel Hill colliery engine, Hazleton.

Pottsville deep shaft engines, Wadesville.

Wyoming Shaft, 2 miles from Wilkes-Barre.

The chief difficulty experienced in using the steam brake seems to be its rapidity and violence of action. When the drum, moving at a high speed, is almost instantaneously checked by the application of the steam brake, the shock is so violent that something is likely to give way—if nothing actually breaks, the strain cannot but be injurious to all the fixed and working parts.

At the Laurel Hill slope this difficulty was overcome by operating the throttle-valve of the steam brake, (See Atlas Plate No. VI,) by a slow motion screw*, so that steam could be admitted very gradually, and the risk of breaking the brake-band was reduced to a minimum. The valve of the donkey engine *f*, which operates the brake, is controlled by a lever placed directly in front of the engineer.

The steam brake used at the Wyoming shaft (made by Mr. Jones, of Wilkes-Barre) is said to have given entire satisfaction.

The engineers state that considerable trouble has been experienced with the steam-brake on the Pottsville engines, but that it is a satisfactory substitute for the hand-brake.

Steam reverse.

In large double link-motion engines the power required to throw the reversing lever over quickly is far greater than can readily be exerted by a man of average strength. Several devices for throwing over the reversing gear by steam have been tried on engines of this class, but they are nearly all open to the objection that a slight pull (or some other motion) is necessary to center the valve of the donkey-engine. To obviate this difficulty the steam reverse shown by Atlas plate No. VI, and in detail by Atlas sheet No. VII, was adopted at the suggestion of Mr. Eckley Coxe, who, I am informed, obtained the device (in a somewhat different form) from a German engineering periodical.

The valve centerer *d* is keyed to a rocking-shaft, which carries a rocking-arm giving motion to the reversing rod *f*; its lower end is connected with the piston-rod by a connecting rod, as shown in the elevation. The reversing lever *c* is connected to the valve centerer at its upper end, being free to rotate on "*2*," but this rotation is limited by the lower end of the lever being attached by a pin "*5*" to a block working between two spiral springs, this motion being limited by the size of the slot "*9*."

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*Not shown on the plate.

The valve rod *l* is connected to the link *g*, which is free to rotate around the center *2*, but is also connected with the hand-lever *c* at "*1*."

The main shaft "*4*" is carried around beneath the link "*9*" in a U bend so that "*3*" is centered in the axis of the shaft.

When the lever is at rest the spiral springs bring "*5*" to the center, and in this position *1*, *2*, *3*, and *5* are in line, and the valve of the reversing engine is centered.

A slight pressure on the lever *c*—say to the right—causes the lever to rock on *2*, compressing the left-hand spiral spring, which allows the lower end of the lever to move about half an inch to the left. By this movement *1* is carried slightly to the right and *3* is carried to the left of the center, moving the valve through its connection with *l* to the left, and admitting steam to the right-hand end of the cylinder.

Immediately upon this pressure on the hand-lever being withdrawn, the springs center *5* and the centers *1*, *2*, *3*, and *5* coming once more into line, bring *3* back to its center, thus centering the valve. The position of the links has no effect on the action of this valve centerer, because the valve is centered whenever the centers *1*, *2*, *3*, and *5* are in line, without reference to the position of the hand-lever or links; and these centers come automatically into line whenever the pressure upon the hand-lever is withdrawn. The action from right to left is precisely similar.

The screws *7* and *8* are set screws to bring the spiral springs into adjustment so that they shall form an exact counterpoise when *5* is centered.

The indicator will be described in the chapter on safety appliances.

The magnificent winding engine at the Pottsville deep shaft, the finest winding plant in the anthracite regions, and probably the most powerful winding engine of this type in America, is shown by Page plate No. 32, which has been prepared from a photograph of the engine.

This plant was designed by Mr. S. B. Whiting, Chief Engineer to the Philadelphia and Reading Coal and Iron company, and was built at the works of Mr. G. W. Snyder & Sons, of Pottsville, Pa.

The steam cylinders are forty-five inches in diameter by five-foot stroke ; the piston-rods are six and a quarter inches in diameter.

The main shaft* (drum shaft) is eighteen inches in diameter with journals thirty-six inches long. The main cranks are set at right angles to each other.

The valves are balanced double-beat, with a play of two inches each, taking motion from a cam. The cam shaft carries a pinion which plays in a rack cut in the upper side of the valve-rod, which moves between slides, as shown by the illustration.

The valve-rod takes its motion (through a connecting rod) from the link block which derives its motion from eccentrics carried on an eccentric shaft five inches in diameter running across in front of the drum. The eccentrics are keyed to the eccentric shaft and give a throw of two and a half inches to the cast-iron box link, which, being supported at the center by pins resting on slides, gives a horizontal ("front and back") motion.

The eccentric shaft is driven by drag cranks from the main crank through connecting rods, and, therefore, has the same motion as the drum shaft.

The cylinders are bolted at the front end to the main bed plate, very much in the same way as in the Laurel Hill engine, Atlas Sheet No. VI. This bed-plate therefore includes the front cylinder head as well as the cross-head guides in one large casting.

Drum.—The drum is a double conical drum fourteen and a half feet in its smaller diameter and twenty feet eight and a half inches in its larger diameter. A twelve-inch brake band of three eighth inch wrought-iron is placed between the two cones. The depth of the shaft is 1576 (?) feet and the cages pass at a depth of nearly nine hundred feet.

These drums were cast, each half in one piece, at the works

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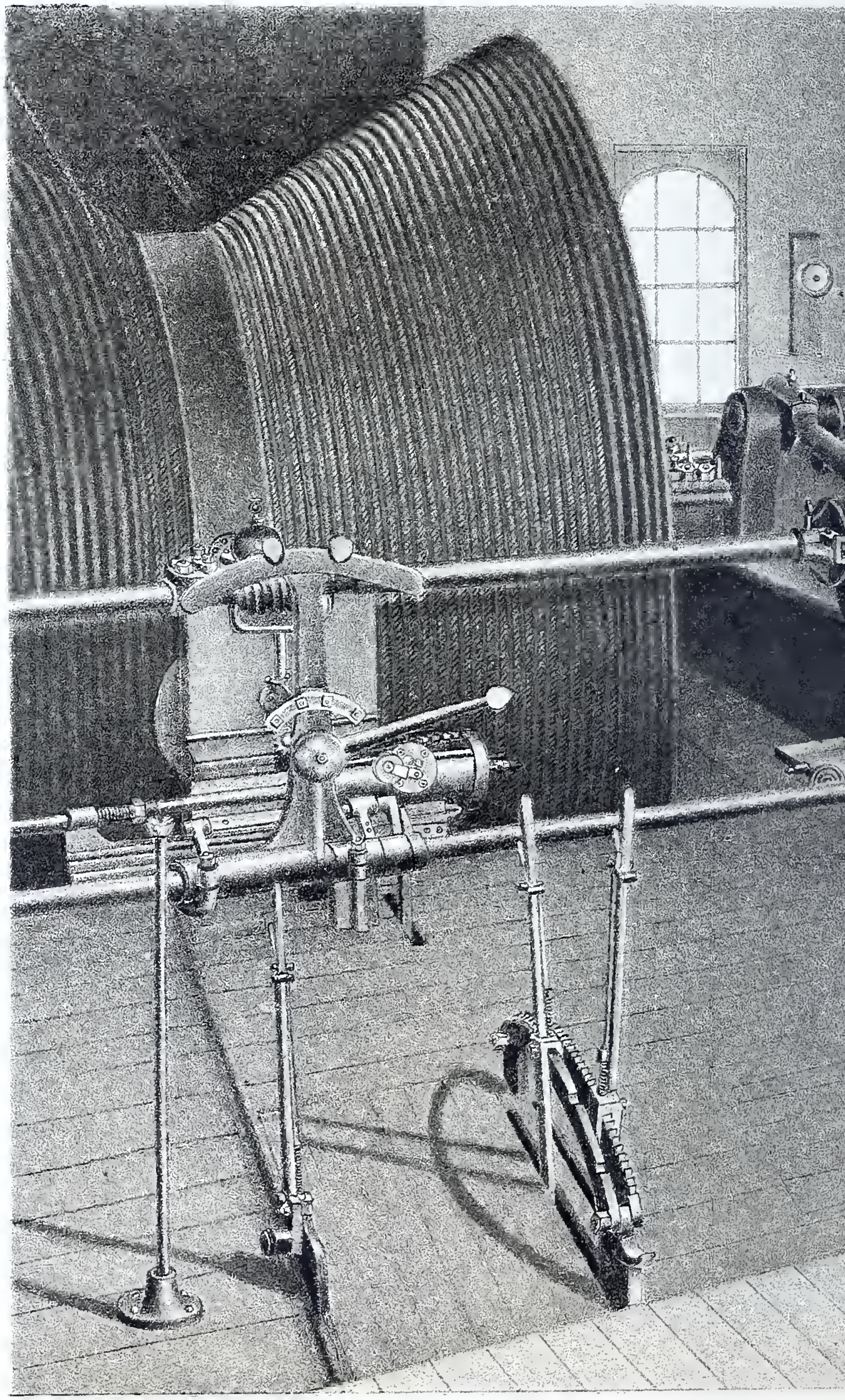
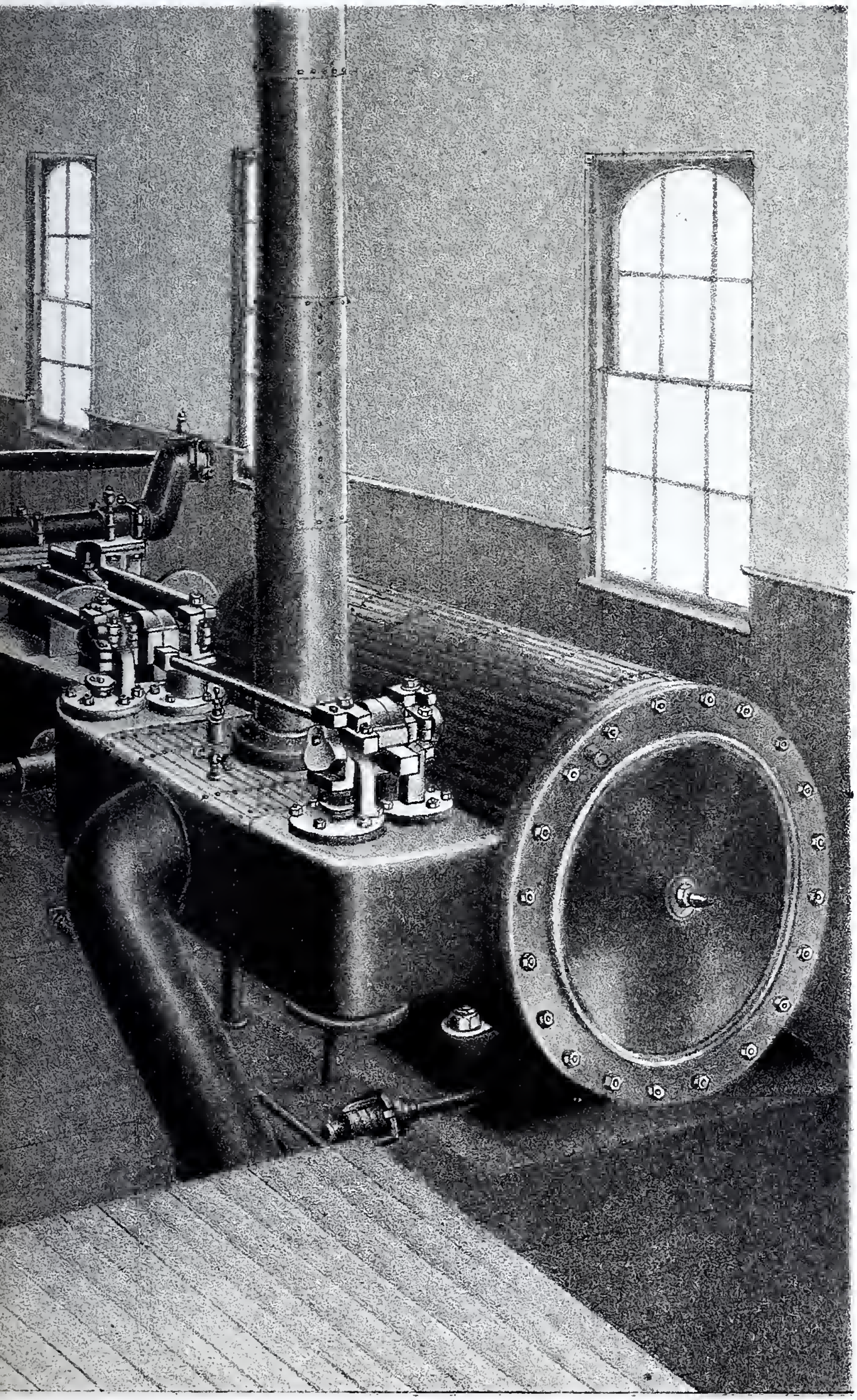
* I am informed this shaft was forged at Krupp's works.



WINDING ENGINE A



TSVILLE DEEP SHAFT

of Mr. G. W. Snyder at Pottsville. This was done to insure perfect continuity of the spiral rope groove. The outer part of the shell is braced by rods (one inch bolts) forming a tension series, in four sets, fastened to rims or hubs on the main shaft, but the middle portion is held by heavy cast-iron spiders through which the tortional strain from the main shaft (and that produced by the brake) is transmitted.

The engraving of the head-frame (Page plate No. 33) shows the proximity of this structure to the engine-house, the lead being little over fifty feet.

For the high speed at which this plant was designed to run, this lead was not found sufficient to prevent the rope from over-running the grooves, and to insure an even coiling of the rope without chafing, travelling sheaves* were placed in front of the drum to guide the rope.

Each sheave is mounted on a carriage travelling on guides inclined parallel to the drum cone (the slots in the engine-house Page plate 33 show this angle). Motion is given to the sheave-carriage by a screw inclined at the same angle, and taking its motion through bevel gearing from the main shaft.

Indicator.—The indicator is an especially ingenious device. As shown by the illustration, the part in sight consists of an indicator arm with a heart-shaped pointer, to denote the position of the cage, but the peculiarity about it is, that its motion is so regulated that as the cage approaches the top or bottom of the shaft the arm moves more rapidly and is very greatly accelerated during the last revolution moving over a space of more than one foot.

This indicator is driven by a worm keyed to the eccentric shaft.

"*Governor.*"—The illustration shows a horizontal cylinder lying beneath the indicator. This is an automatic governor by which the steam is shut off and the steam brake applied to stop the engine whenever the speed exceeds a certain limit fixed upon as the maximum speed at which the engines can safely run. It was especially designed to decrease the risk of the engine running away, whether from accident, carelessness or breakage. It works on the principle of the cataract. I have never seen this appliance in operation.

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*Called a "fleet-gear."

Overwinding.—There is also an arrangement by which the steam can be automatically cut off at any point, and the steam brake applied to prevent the possibility of overwinding.

Steam brake.—The steam brake is so arranged that the steam merely acts as an auxiliary to the power exerted by the engineer on the brake-lever. The load is so perfectly balanced that the use of the steam cylinder is not actually necessary, as the brake is rarely applied to stop the engines and drum, but is generally applied when the engine is at rest, to hold the drum and prevent its being accidentally started.

Steam reverse.—The engineer is aided in throwing over the valve-gear by a donkey engine located beneath the floor of the engine room.

A rather peculiar method of operating the hoisting gear is in use by the Delaware and Hudson coal company, at several of their largest shaft collieries. At the Plymouth No. 3 shaft the engine is a double second-motion engine, (geared 3 to 1) the driving shaft carrying two driving pinions. There are two drums, one for each shaft, which may be coupled by clutches to run together.

The peculiar facilities afforded by such an arrangement for winding from either compartment, or from any landing, are obvious; but the increased first cost of plants built on this plan, the increased risk of overwinding, etc., the lack of counterpoise, the unequal and varying strains to which the engine, gearing, drums, rope and sheaves are subjected, and the greater frequency and increased cost of repairs, will probably prevent its more general adoption.

At the Plymouth shaft the cylinders are sixteen inches in diameter by thirty-six inch stroke; the drums are nine feet in diameter and are each provided with a hand-brake; the ropes are one and a quarter inches in diameter. The shaft is 336 feet deep.

Friction Gear.

Friction gear is seldom employed on winding plants at large collieries, but the use of the cone and other friction gears for operating dirt-planes, is very common.

At the Cross Creek colliery No. 2, at which coal is raised simultaneously through a double slope from different levels, the winding is effected by the friction gear introduced by Mr. Coxe, which has recently been adopted at other works, and with some modifications, is being applied to winding machinery for ore mining.*

The drums both run free on the main driving shaft, and the power is transmitted by a series of friction blocks which bear against the inside of a rim forming part of the drum frame. The friction blocks form a complete circle almost as large as the inside diameter of the drum. They are held in shoes somewhat resembling a brake block holder, toggle jointed to a rod which is attached in like manner to a movable sleeve on the driving shaft; this sleeve runs with the driving shaft. The whole device resembles a dished wheel in which the sleeve is the hub, the rods, the spokes, the friction blocks forming the circumference.

When the sleeve is forced in towards the drum, the wheel becomes less dished than before, its diameter is increased, the friction blocks take hold, and the drum revolves with the shaft.

The *descent* of the car can be controlled either by a brake or by making the drum work against the friction blocks.

I expected to obtain an accurate description of this device from Mr. Coxe, but have not received it as this goes to press. The above imperfect description is compiled from memory

Drums.

Conical drums are rapidly coming into general favor for shaft collieries, but as yet only a few have been built, and cylindrical drums are still in general use for winding throughout the anthracite region, both at shaft and slope collieries.

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*A similar winding plant was built for Mr. Edwin Mickley, mining superintendent of the Thomas Iron company at the company's shops in 1882.

Cylinder drums of moderate size are commonly constructed of heavy cast-iron spiders with six or eight arms, surrounded by a lagging of timber six or eight inches thick. When very large the spiders are cast in two or more pieces and bolted together. Very heavy spiders are always necessary, and their number depends not so much upon the size of the drum as upon the number of coils the rope makes upon it.

Drums driven by geared engines often have the spur wheel placed in the center, as shown by Fig. 46. This has been considered to be one of the best plans for geared engines, as it reduces and localizes the tortional strain to which the drum shaft is subjected.

When the center of the drum is not occupied by this pinion, this location is frequently chosen for the brake. This latter arrangement is probably better than the former, and the center of the drum seems to be an especially favorable position for the brake; and it seems doubtful whether it is not better to gear the driving-pinion to a cog-wheel placed at the side of the drum, or on the drum shaft entirely independent of the drum, and to make the drum shaft strong enough to bear any tortional strain to which it may be subjected.

Some very large cylindrical drums are in use, even larger than circumstances seem to require. Those at the Kohinoor colliery (two on one shaft) are twenty feet in diameter by little over two feet in breadth.

Probably more than three fourths of the total number of drums in use at shaft collieries are between eight and sixteen feet in diameter. There are very few large shaft collieries at which the drums are less than ten feet, and the majority of such workings are furnished with drums twelve feet and upwards in diameter. Drums at slope collieries usually range from eight to twelve feet in diameter.

Atlas sheet No. VI shows in detail the drum at the Laurel Hill colliery. Four heavy cast-iron centers, strengthened by twelve ribs, are keyed to the drum shaft, and to each of these centers twelve heavy wrought-iron arms are bolted, which carry the angle plates (made in segments) to which the lagging is bolted. This is probably one of the best methods of constructing large cylinder drums for very heavy work.

Conical drums are as yet principally confined to shaft collieries in the Wyoming district. While the engineers throughout the region, with few exceptions, believe that conical drums should be adopted at all large collieries at which the coal is raised from a considerable depth, the comparatively small cost of cylindrical drums as compared with that of conical drums, the impossibility of determining the proper shape of the cone before the exact depth of the shaft is known, and the time then consumed in the manufacture of the conical drum, have induced many engineers, while admitting the superiority of the conical form, to adhere to the old style cylinder drum.

Brakes.

Wrought-iron brake bands mounted as shown by Altas Plates Nos. III and VI, are used to the almost entire exclusion of other forms. The levers for transmitting the power from the hand-lever or treadle to the brake band, are variously arranged. In some cases the force is multiplied by several short levers, in others one long lever is used.

The treadle is rapidly being replaced by the hand-lever, and is now seldom seen except at old collieries.

A single brake band entirely surrounding the drum is sometimes used, but the necessity of providing springs or counterweights to prevent it from resting on the drum, and its tendency to oscillate, are serious objections; for very small drums it makes a perfectly satisfactory brake.

Brakes operated by a small winch and chain—the chain being attached to a long lever or series of levers—are common enough at old collieries, and have been adopted at some recently opened works. They have been most largely used on self-acting planes, to the requirements of which they seem admirably adapted.

Brake blocks are rarely used. While better results can doubtless be obtained with brake blocks (when properly made to give the highest coefficient of friction) than with iron bands, the latter are probably safer and less likely to get out of order and work badly. This subject is ably discussed in Percy's recent work on the "Mechanical Engineering of Collieries."

CHAPTER XV.

Winding Machinery and Appliances.

We may distinguish three separate classes into which all the various forms of head-frames now seen at anthracite collieries may be divided.

The majority of head-frames in the Wyoming basin, where shaft collieries are most common, are of the square upright pattern, [See Atlas Plates IV and XX for illustrations of the Exeter and Oakwood shaft head-frames,] with or without inclined braces.

An example of an upright frame with inclined braces is shown by Atlas Sheet No. XVI. This form may be considered a compromise between the two elementary forms shown by Figs. 44 and 45.

The triangular form is in use principally in the English coal fields,—in the last few years has been adopted by the Philadelphia and Reading Coal and Iron company at a number of collieries.

In the illustration D is the drum, C the sheaves, on one of which the force aD acts, on the other the force a'D, being the pull towards the drum,—the two vertical forces

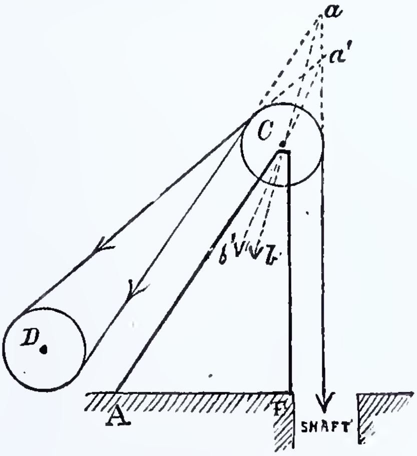


Fig. 44.

are approximately equal to these. There are, therefore, two resultants, *ab* and *a′ b′*, the direction of which is determined by lines from *a* and *a′* through the center of the sheave C.

It is evident that the most simple structure of maximum stability will have a vertical limb parallel to the vertical forces, and an inclined limb approximately parallel to a line joining the centers C and D, but as it is not usually feasible to make AC parallel CD, the inclined limb is generally given less batter. While this is apparently the principle upon which all triangular head-frames should be built, other considerations frequently render the adoption of an upright frame advisable.

Page plate No. 33 shows what is probably the finest example of this style of head-frame in America. It is built of Phœnix iron columns, and I understand was guaranteed by the builders to stand a strain of one hundred and fifty tons.

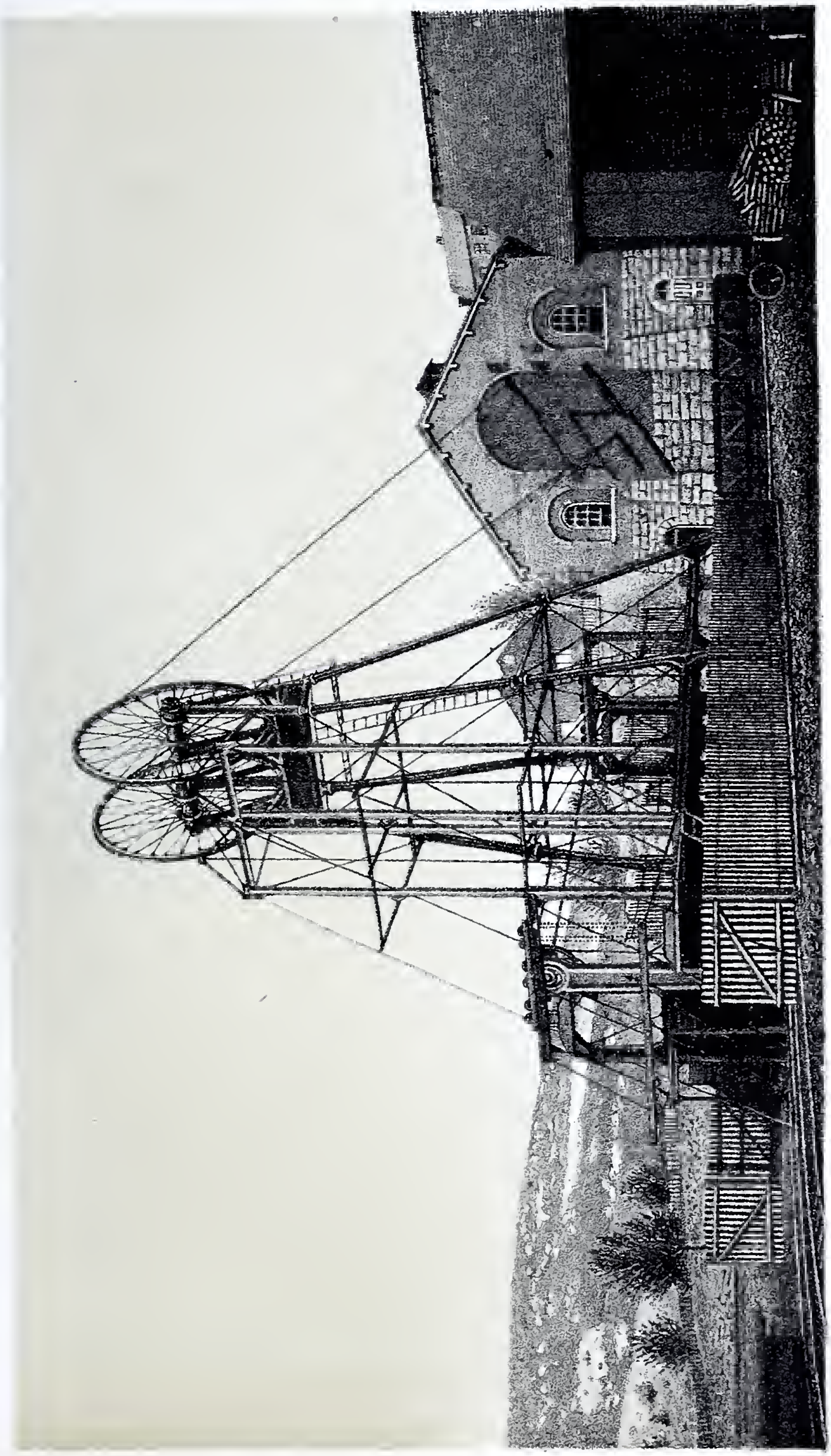
The structure consists of four main columns, each column composed of four segments riveted together after the Phœnix pattern, braced by horizontal braces and diagonal tie rods. The pillow blocks and foot blocks are of cast-iron, the latter being securely bolted to a heavy foundation of masonry.

Vertical height of the head-frame to the center of the sheave journals, fifty feet. The sheaves are tension wheels sixteen feet in diameter, with cast-iron rims and cast-iron double hubs.

It will be observed that, to obviate the necessity of a central frame to support the inside pedestals, the sheaves are set in a heavy frame resting on the four main columns.

The necessity of adding this top pedestal frame to all triangular head-frames, and the cost of building an additional framing to carry the cage guides, are disadvantages which, in the opinion of many engineers, more than equal the advantages of triangular head-frames.

When the square upright head-frame is adopted, it is usually necessary to fasten the foot blocks on one side of the frame very securely, by bolting them down to a massive



POTTSVILLE DEEP SHAFT.

foundation to neutralize all tendency to rotation on the leg nearest the drum. (See Atlas Sheet No. IV.)

If this is not done the frame is usually stiffened (if built of timber) by an inclined brace as shown by the dotted line AB in Fig. 45. In this frame the resultant forces acting in the direction of the arrows *ab* and *a′b′*, tend to cause rotation around B.

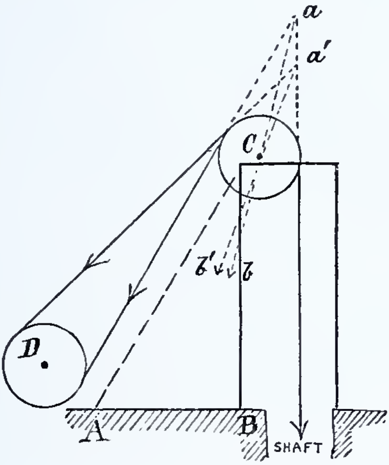


Fig. 45.

The upright frame, braced by an inclined leg, is probably the most common form in use, especially at collieries where the head-frame and breaker are united in one structure; but when square iron frames are used, the structure can then be securely held by the foot-plate bolts, and this method of bracing is not necessary.

When the head-frame is built as an individual structure, entirely independent of the breaker, it is usually made from thirty to fifty feet high. With direct acting engines the height of the head-frame should be sufficient to allow *at least two thirds of a revolution* play, between landing the cage and the overwinding point, and at a majority of shaft collieries the margin is considerably greater.

But when the breaker and head-frame are combined in one structure, the head-frame is seldom carried up more than thirty feet (and often much less) above the landing at the top. When the height of the rope-socket above the cage floor is deducted from this, it will be seen that with first-motion engines and comparatively large drums the margin of safety against overwinding is reduced to one half or even one third of a revolution or less.

Guides.

Wire-rope guides or "*conductors*" have been largely used in English mining districts, but are not used in the anthracite regions. The most enthusiastic friends of wire-rope conductors, do not claim for them any marked superiority over fixed wooden guides, in steadying the cage or in securing immunity from accident; but in circular shafts of small cross-section, used for both winding and ventilation, wire-rope guides are often adopted because the shaft is then left entirely unobstructed, while with wooden guides heavy buntons are needed, which greatly diminish the effective ventilating area of the shaft.

The guides are usually about six (6 to 8) inches wide and from four to six inches thick, fastened by counter-sunk bolts to the buntons, in which gains are sometimes cut to receive the guides.

At the landing their width is made somewhat less than in the shaft, so that the safety catches will not take hold and jamb fast when the cage is landed. This precaution is absolutely necessary only at collieries provided with telescopic cage rests, ("keeps") on which the cage gradually settles to a bearing level with the landing rails.

Cage rests, Keeps or Wings.—These do not differ materially from those used at English collieries.

Telescopic cage rests have been introduced at a few collieries, (*e. g.* Pottsville deep shaft) but it is not likely that they will come into general use except at very large collieries. Those at the Pottsville shaft are of the plunger pattern, working in an oil cylinder, on the cataract principle. The weight of the cage and car forces the plungers home, and they are thrown out, when the cage is lifted, by a spiral spring inside the cylinder.

Atlas Plate No. IV shows the arrangement of the wings, and the levers by which they are made to work together at the Exeter shaft.

At slope collieries using a slope carriage, the wings are arranged in a similar manner.

The wings* are sometimes made of iron, sometimes of wood. A timber frame is probably preferable when no provision is made by springs, etc., to break the shock caused by the cage settling on the wings.

Stops.—Various forms of stops are used to prevent the mine cars on the empty track from running (down grade) into the shaft. Some are automatically operated by the cage as it settles to a bearing on the wings, others are self-acting and must be opened by a lever before the car can run upon the cage.

They usually consist of a block of wood or a small iron bar projecting over the track on one or both sides, or a bar bent at an obtuse angle and pivoted on one side of the track, which acts as a ratchet or escapement, and holds the car until it is thrown back to let the car pass. The hind wheel of the car throws the stop over and blocks the track so that the next car cannot pass.

Sometimes these stops are replaced by a heavy beam or gate, which is lifted by the cage, and drops back into position when the cage descends. A description of these devices properly belongs to the chapter on Safety Appliances.

Cages.

Single deck cages for raising one mine car at each hoist are used to the almost entire exclusion of other forms. The Pottsville shaft plant was designed for double deck cages, and an ingenious automatic appliance for raising and lowering the cars from the upper platform was built, to obviate the necessity of decking the cage twice at each winding.

Two double-deck dummy cages are placed in front and two behind the two winding apartments, and these cages are connected by chain gearing in such a way that the loaded cars from the shaft cage, when run upon one of these cages, raise the cage on the *opposite side* containing an empty car on the upper deck; the dummy cage on the *same side* of the shaft is at the same time raised, and the cage on the opposite side (from which the empties were taken to load the shaft cage) sinks.

---

*The term commonly applied to cage rests in the Wyoming Valley.

Double deck cages are common enough in European mining districts, but in anthracite mining practice the great weight of the cars and of the coal they carry, the comparatively shallow depth of the workings, and the large size of our shafts, (sometimes allowing space for four winding compartments,) make them unnecessary.

For deep winding of large quantities of coal from shafts of comparatively small cross-sectional area, double deck cages may be preferable to the ordinary single cage; but in this, as in many other particulars, our mining engineers will be very slow in following the lead of English colliery viewers.

Iron cages are largely used throughout the anthracite regions; but from the experience of a large number of mining engineers, I judge that cages constructed of wood, stiffly braced with iron bolts, give more satisfactory service than those built entirely of iron. Iron cages are very difficult to repair; they sometimes become jambed fast in a shaft, and much time is lost in taking them apart. A cage of wood and iron is readily taken apart in the shaft, (in case of accident from rope breakage, etc.,) and may be quickly repaired; it is more elastic, and better able to stand without injury the jars and shocks to which all cages are subjected.

The use of self-dumping cages is principally limited to cages used for simply raising mine cars from the surface to the top of the breaker.

Few mining engineers consider the use of self-dumping cages for winding shafts advisable or even allowable. Self-dumping cages are at best cumbersome, more or less complicated, and, consequently, liable to accident.

An iron self-dumping cage, used by the Lehigh Coal and Navigation company, is shown by Fig. 3, Atlas sheet No. XI, which will serve to show the method of constructing iron cages. They are built of plate-iron and riveted throughout.

Atlas sheet No. XI also shows two forms of shaft cages or carriages, one in use at the Audenreid shaft (Fig. 1) of the Lehigh and Wilkes-Barre company, and one designed by Mr. J. H. Bowden for the Susquehanna Coal company.

As dumping cages are principally used for raising cars from the surface to the top of the breaker, they will be discussed in the chapter describing the coal breaker.

All shaft cages are now provided with a bonnet to protect them from objects falling down the shaft. The bonnet is commonly made of heavy wrought iron plate.

The law requires all cages to be provided with a safety catch (see chapter on Safety Appliances) and bridle chains.

All cages are provided with some contrivance for holding the car firmly in place while the cage is in motion. Some of these catches are automatic, but some must be set by a hand-lever.

One form of catch used on slope cages is shown by Fig. 43; another method of holding the car is shown by Page plate No. 36.

The catches on shaft cages consist either of a lever with a weight or spring at one end, and a recess, forming a hook at the other, to receive and hold the front axle of the car; or of an iron plate which falls into place between the wheels of the car; or of devices similar to that shown by Page plate No. 36, in which the platform of the cage consists of two parts, one fixed and the other movable, each carrying a part of the track. When the cage is raised from the keeps, the movable part on which the car stands remains stationary while the cage rises six or eight inches, and the car-wheels settle into this space. Sometimes the cage is so made that the car settles down until the bed-frame rests directly on the rails.

This latter arrangement is preferred for dumping cages, but as it weakens the cage and adds to its weight, it is seldom used on shaft cages. (See Atlas plate No. XI.)

A plate or block, mounted on elbow-joints on the outside of the track and arranged to automatically fall in between the wheels, makes an excellent catch.

The length and breadth of the cage is governed by the size of the mine car. Recently opened shaft collieries are provided with cages varying from ten to nearly twelve feet in length; in breadth they range from five and a half to seven feet. Their weight varies with the size and capacity of the mine car and method of construction; ranging from one to two tons.

Barneys.—The use of the barney is by no means limited to outside planes, but as yet they have been used at comparatively few slope collieries.

It is evident that they are not admissible where several landings are worked.

At collieries working only one lift, or where the slope is sunk at once to the bottom of the basin, and higher lifts worked by counter gangways, the barney may be advantageously adopted.

Plans for the arrangement of the barney pit at the bottom are shown by Atlas sheets Nos. VIII and IX. The former shows a series of automatic latches by which the barney is made to run down into a pit, pass under the loaded car, and come up behind it.

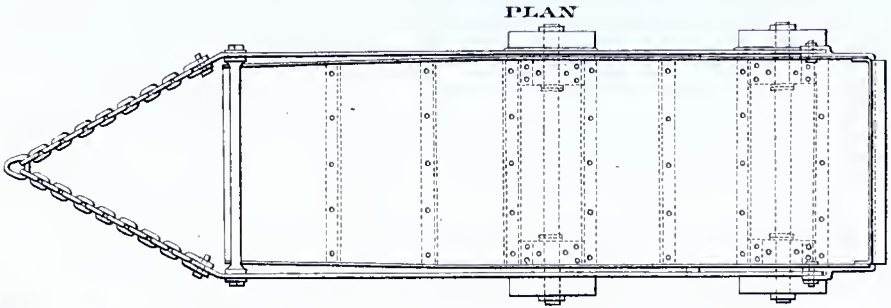
The first latch is held open by a counter-weight, but is closed by the barney passing up over it with a loaded car, thus completing the continuity of the upper track. The operation of the second latch is sufficiently obvious. The empty cars are taken off over a bridge.

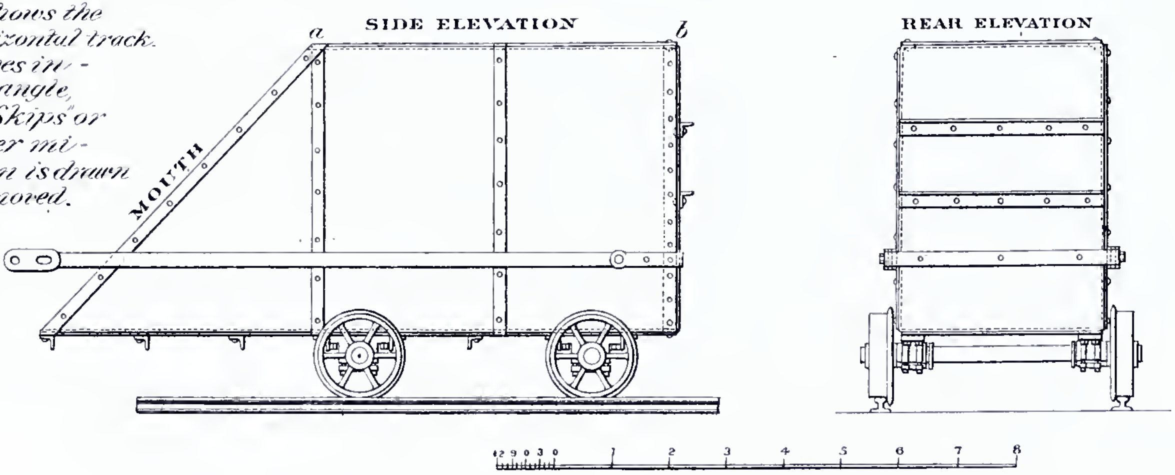
While the use of a barney in winding, saves the labor of attaching the rope to the cars, and also of detaching it at the top, its additional weight, the necessity for a double line of rails, truck-pit, etc., have prevented its adoption at many slopes admirably suited to this method of winding.

At the Cross Creek colliery, Mr. Eckley Coxe has in use a barney arranged to dump the cars at the top of the breaker. This is effected by constructing the barney of two frames, one lying on top of the other; the top frame carries the bumpers and is attached to the truck bed-frame at the rear end by a massive hinge.

When the car reaches the top, the front wheels run over a knuckle into chairs, the top frame of the barney (as the latter slowly rises) catches beneath the car frame and rises

"GUNBOAT"
BUILT AT
THE HAZLETON SHOPS
FOR
COXE BROS.' COLLIERIES.





Note: This drawing shows the Gunboat lying on a horizontal track. It is used only on slopes inclined at considerable angle, and is similar to the "Skips" or "Monitors" in use in other mining districts. The plan is drawn with the top (side) ab removed.

until it is nearly vertical, lifting the rear end of the car so high above the track that the coal runs out of the front end into the dump-shute. After the car is dumped the barney is lowered a few feet, its upper half settles down upon the truck frame, and the car falls back into place.

Gunboat.—This name is applied to a self-dumping car used for raising coal on slopes. It resembles a skip,—but runs on wheels instead of between guides,—being open at one end only; is usually built of boiler iron, and made to hold as much as one or two mine cars. Page plate No. 34 shows a gunboat built at the Hazleton shops for the Messrs. Coxe Bros. It has a capacity of more than one hundred and fifty cubic feet when loaded even, holding therefore about four tons of coal.

It will be observed that the wheels are fixed (tight) upon the axles, which are seated in pedestals. This plan is peculiarly adapted to gunboats, and is also used to some extent on mine cars. It has some of the advantages of both the loose wheel and fixed axle, and the fixed inside wheel, among which the following may be mentioned: 1st. Less risk of bent axles than with inside wheels; 2d. Less wear of journals, and more regular running than with loose outside wheels. Objections: not readily lubricated and the pedestal is difficult of access.

The bottom and back are both stiffened by angle iron.

An improved gunboat, recently built for the Lehigh Coal and Navigation company's No. 12 colliery at the company's shops, is shown by Page plate No. 35. The back is stiffened and protected by three-inch plank, backed by three five-by-seven timbers, and the weight is more evenly distributed by the V straps than by a single strap.

A drawing showing the details of the journal bearings is also shown. A rubber cushion is placed in the space ruled with vertical lines, to diminish the jar from shocks and to decrease to a minimum the danger of derailment, while the whole bearing is arranged to allow of a slight oscillation to prevent the box from being broken by the wrenching and swaying of the whole body of the gunboat when loaded with coal.

In the upper right hand corner of the plate is a sketch showing the gunboat dumping coal at the top of the breaker.

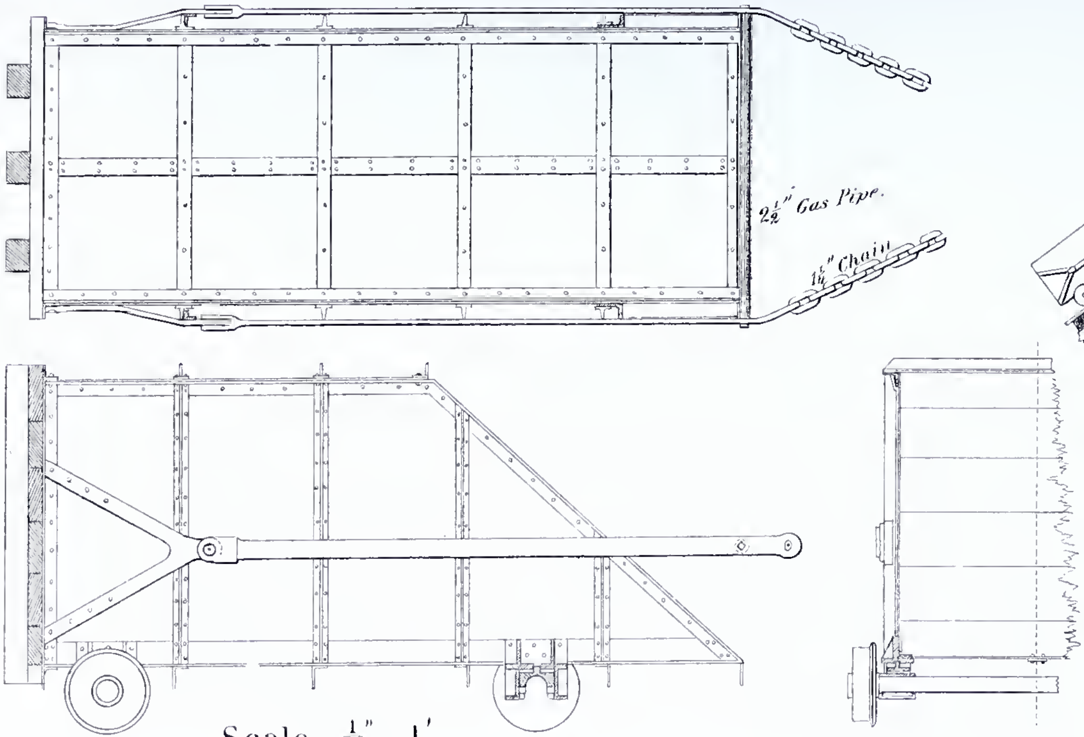
The dumping is sometimes accomplished by placing a small wheel on each side near the middle of the body of the car. These wheels strike an auxilliary track at the top of the breaker, just before the front wheels run over the knuckle. The same object is also attained by making the hind wheels of very much broader tread (about double) than the front wheels, and from a point three or four feet below the knuckle, laying a second track outside the main track extending upwards in continuation with the line of slope for a distance of about fifteen or eighteen feet above the knuckle. The front wheels then pass in over the knuckle *between* the outside rails, but the hind wheels being of broader gauge take a bearing on the outside track and run up above the knuckle.

As the mine cars must be tipped in the mine when a gunboat is used, the economy effected is not due to the gunboats' self-dumping property, but is limited to the labor saved in switching, running the cars on and off the slope, attaching and detaching the rope at the bottom, and the same operations at the top of the breaker. We may assume that the labor of tipping (dumping) the mine cars inside is about equal to that of dumping them at the top of the breaker when a gunboat is not used. Hence, at a colliery of average capacity raising coal from a single level, the practical result is the abolition of the labor of from two to four men.

As an offset to this, we have a variable increase in the percentage of culm due to double handling of the coal, resulting in a loss that may or may not be greater than the gain from labor saved. The amount of this breakage depends on the physical character of the coal.

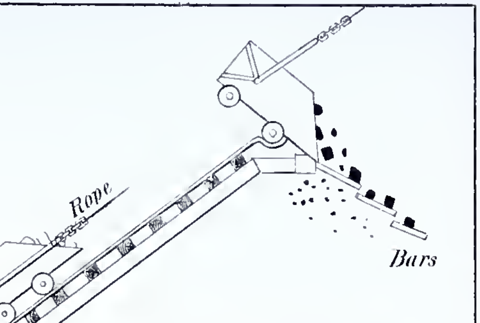
It should not be forgotten that the life of the mine cars is greatly prolonged by the use of a gunboat; the annual expenditure for repairs is greatly reduced, and somewhat less power required for winding.

Gunboats are also used for raising refuse matter (culm, slate, and rock,) on dirt planes. The cost of handling this

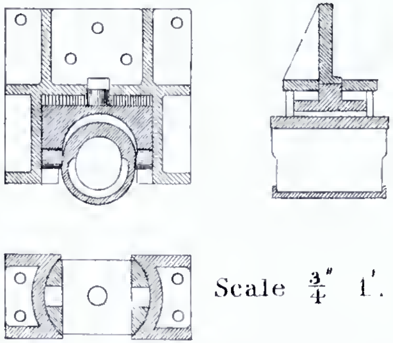


Scale ¼" 1'.

GUNBOAT FOR COLLIERY Nº 12.
LEHIGH COAL AND NAVIGATION COMPANY.



Gunboat dumping Coal



Scale ¾" 1'.

Inproved Pedestal
and Bearings.

material might be greatly reduced by so locating the foot of the dirt plane that the gunboats could be loaded either directly, or through shutes, from the culm, slate, and rock pockets in the breaker. Gunboats ("Monitors") are used on the high dirt plane at Shamokin ; but the refuse is brought to the foot of the plane in dumpers. The monitor dumps the material into a pocket at the head of the plane, from which it is loaded into dumpers and hauled to the end of the bank,—a distance of half a mile,—by a locomotive.

At present gunboats are used at only a comparatively small number of collieries. Some of the most able mining engineers in the region hold them in high esteem, and others are coming to admit their merits ; but among the majority there is still a firmly rooted feeling against the adoption of any system that involves double handling (dumping) of the coal.

The gunboat shown by Page plate No. 35 has a capacity, when loaded even with the mouth, of about two hundred and fifty cubic feet, holding, therefore, about six and two third tons.

Slope carriages.

When the slope inclination exceeds thirty-five degrees the cars are usually raised on slope cages. The coal may be prevented from falling out on dips of thirty to somewhat over forty degrees by building the cars with check beams across the top, as shown by Page plate No. 29, care being taken not to load the car quite full.

Slope cages are often built to run on a slope track as arranged for mine cars, but to insure stability the gauge is generally broader than the gauges used for mine tracks. The height of slopes designed for slope cages is frequently double that of ordinary slopes. The headroom necessary is governed not so much by the form of slope cage as by the length of the car, and the inclination of the seam. This height is less when the cars are placed on the cage with their length *across* the slope than when they are run on lengthwise ; but as this arrangement increases the *width* of the slope, it is often no better than the other. When this plan is adopted, the cages are sometimes made to pass at a turnout, so that at the center of the slope only, is the width sufficient to accommodate two cars placed lengthwise. This width (at a minimum) for ten-foot cars is about twenty-four feet in the clear.

When the inclination is very steep the wheels are sometimes placed on the sides of the cage above the center of gravity, and run on tracks,—which resemble guides in their action,—supported by timbers on each side of the slope.

Such slopes resemble shafts in many particulars, and in metalliferous mining districts would be so described. They frequently require timbering similar to shaft timbering; their cages resemble shaft cages, but the slides are replaced by wheels, and the guides by rails, which may be wooden; the platform on which the car rests, forms the bottom instead of the top of the cage, and the rope is attached to the framing overhead.

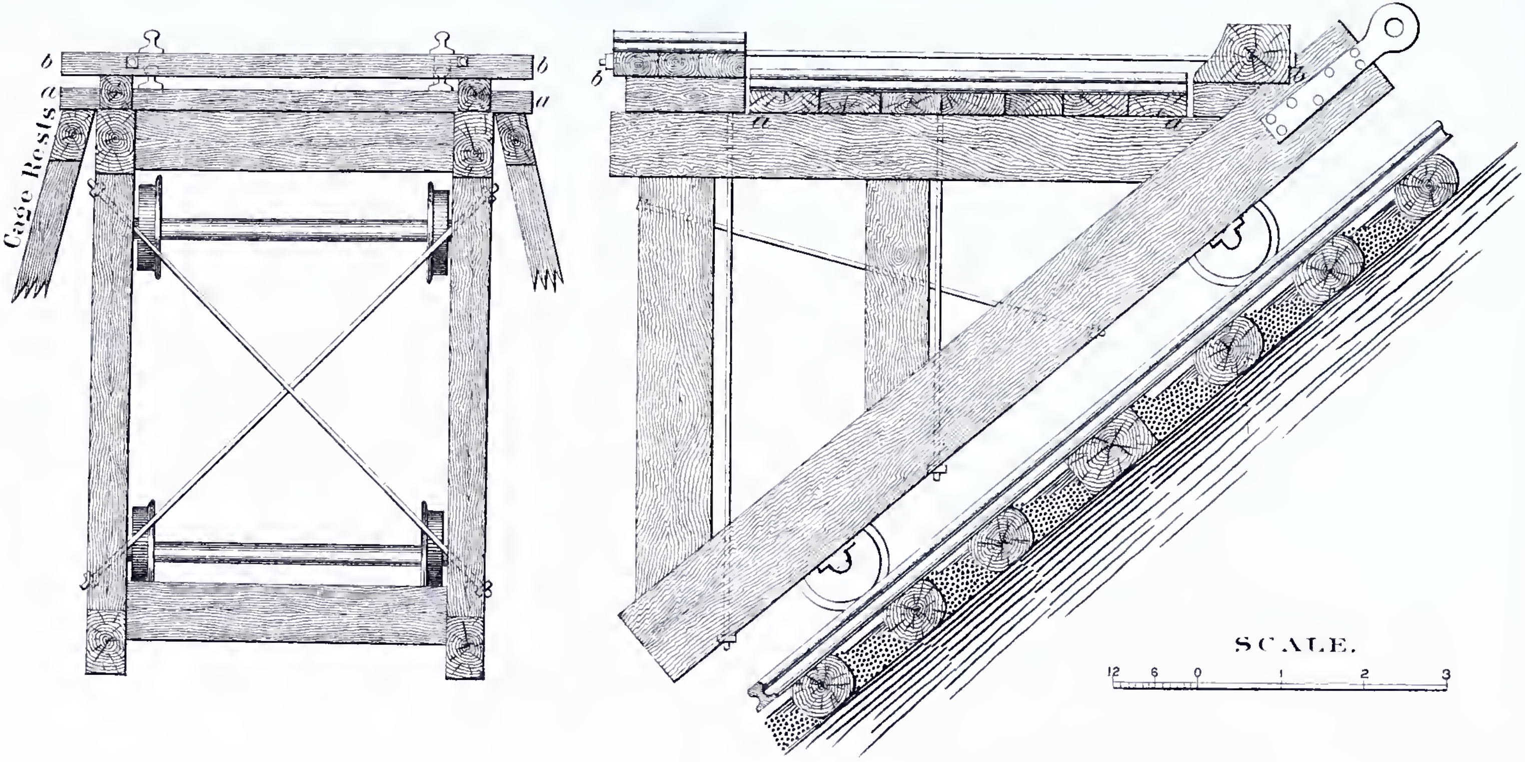
The gunboat seems peculiarly adapted to raising coal on steep dips, and will doubtless replace the slope carriage at many such collieries. Several different forms adapted to steep dips have been in use for some years in the Mahanoy district with most satisfactory results.

Car locks similar to those used on shaft cages are also applied to slope cages. Page plate No. 36 illustrates one method of holding the car securely on the carriage. The platform *a a* is movable. When the cage settles upon the keeps (cage rests) this platform is caught and held while the whole cage settles down until *b b* rests upon the keeps: the track on *a a* is then at the same level with that on *b b* and the car is run off and another run upon the carriage.

Another plan for automatically locking the cars is shown by Fig. 47. The draw-hooks are made to have a play of six or eight inches. When the cage settles upon the keeps and the rope becomes slack the draw-hooks drop by their own weight, carrying down the segment of rail to a level with the cage track, when the car may be run off.

Another car is then run on and when the rope tightens the draw-hooks slide up (until they come to a bearing against a stop) carrying the rail up between the wheels of the car, as shown by the illustration.

SLOPE CARRIAGE, MAHANOY CITY.



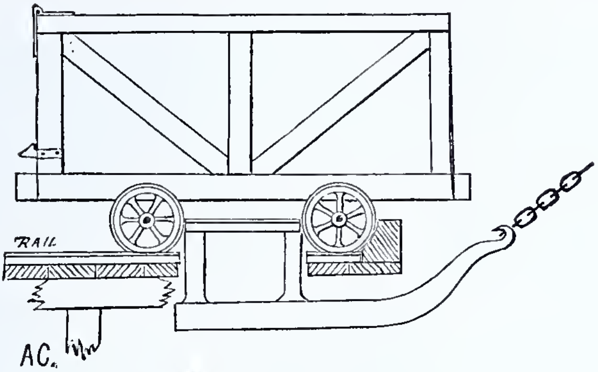


Fig. 47.

Single-Rope winding plants.—The self-acting plane, shown by Atlas sheet No. X, has already been briefly described, but as it is allied to this method, further consideration of its peculiarities properly belong to this chapter.

On planes or slopes of great length the use of a single rope has several undeniable advantages, but it is also open to several objections.

Advantages.	*Disadvantages.*
Only one rope is required.	When this one rope breaks, no coal can be raised or lowered until a new rope is obtained, (or the rope is spliced.)
It does away with large, cumbersome drums, and the chafing of the rope as it coils upon the drum.	The rope is bent around sheaves *five* times (in each case described) producing *five times* the wear and tear of a drum, and making spliced ropes unsafe. It is also bent in several different directions.
The rope is always in line with the knuckle and bearing pulleys.	
The rope wears evenly throughout its length, but the wear will nevertheless be greatest at the rope sockets.	When the ends near the rope sockets wear out and a piece is cut off, the tension sheave (Col. Brown's) is adjusted to compensate for this shortening, but as this can only be done a few times, the chains must be lengthened or the rope discarded.
If the cars get off, the rope may slip and prevent kinking.	When the empty car is detached the rope may slip.
	In the double sheave plan, (Plate X) the rope or machinery may be broken by unequal binding on the sheaves.

The plan devised by Col. D. P. Brown, of Lost Creek, for No. 3. colliery (Lehigh Valley Coal Co.) is roughly shown by Fig. 46. An old drum with two grooves is used in place of two driving sheaves. The sheaves nearest the drum are placed higher than as shown by this sketch, and the diameter of the tension sheave is larger. The parts are distorted in order to clearly show the course of the rope around the sheaves.

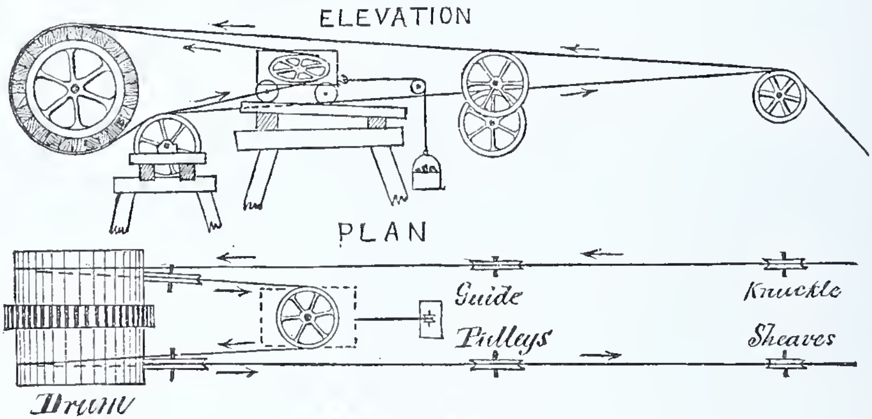


Fig. 46.

It will be observed that the rope runs over the drum twice (one three-quarter turn on each side) and around the tension and contact (two) sheaves at each winding. As these three sheaves are only about one half the diameter of the driving sheaves (drum), the wear on each sheave is twice as great as that produced by the driving sheaves.

Taking the wear of the rope on a drum of this diameter, caused by one winding on one side, as a unit, we have as the comparative wear:

Winding with two ropes.

Rope coiling on drum, (loaded side,)	1
Rope uncoiling from drum, (empty side,)	1
Total for drum,	2

Winding with a single rope.

Rope bending on driving sheave,	1
Rope unbending from driving sheave,	1
Rope bending on contact sheave,	2

Rope unbending from contact sheave,	2
Rope bending on tension sheave,	2
Rope unbending from tension sheave,	2
Rope bending on contact sheave,	2
Rope unbending from contact sheave,	2
Rope bending on driving sheave,	1
Rope unbending from driving sheave,	1
Total for single rope plan,	16

Or eight times the wear caused by a drum. This ratio should be reduced by a certain allowance for the absence of the wear caused by the coils crowding together on the drum.

As the wear on the drums and sheaves* is 39 *per cent.* of the total wear on slopes, which we may divide into 13 *per cent.* for the drum and 26 *per cent.* for the sheaves, on slopes averaging 933 feet in length, we may assume for this length of 1400 feet that the slope wear (bearing pulleys, etc.) will be double that, reducing the relative *per cent.* of wear on drums to 6½ per cent.—the wear on such a slope due to drums will therefore be as 6½ is to 61 (the wear on the slope); but if the single rope be adopted this 6½ is increased nearly eight times—say six times—making this wear equal to 39. Adding to this the wear due to knuckle sheaves (26) and the wear on slope (61) we have 126 : 100, or about *25 per cent.* increased wear of the rope due to the substitution of this method of winding. This is the minimum—the maximum increase in wear might reach 50 *per cent.*

Wire Ropes.

Iron wire ropes were first successfully used in the Hartz mountain mining region about 1836.

The principal advantages of iron or steel wire over hemp rope, are:

1. Greater durability.
2. Less weight for equal strength.
3. Reduced cost per ton of material raised.

The same essential advantages can also be claimed over chains or flat bands of iron or steel.

A series of figures given by Taylor (Statistics of Coal) show that the cost per ton of raising material with hemp rope is about three times as much as when wire rope is used: the data furnished show an average cost of $1\frac{30}{100}$ cents per ton in the former and only $\frac{36}{100}$ cents per ton in the latter case. These observations apply to shaft collieries only; at slope collieries the difference would be much greater on account of the great frictional wear on the bearing pulleys.

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*See "Wire ropes" below, three or four pages further on.

In the anthracite regions the cost of rope per ton of material raised is subject to great variations, which depend principally upon the conditions enumerated below.

1. The depth.
2. Working load.
3. Diameter of sheaves, drums, and pulleys.

The cost at slope collieries is much greater than at shaft collieries.

The following table shows the tonnage raised by twenty-three wire ropes used at eleven different slopes on dips ranging from fifteen to sixty degrees; and the tonnage raised by six ropes at three shaft collieries in the anthracite region. The cost per ton is estimated from Roebling's price list for October, 1880.

The cost per ton per hundred feet of lift is seen to vary between .029 and .161 cents, but the latter figure is evidently exceptional,—the rope was probably of inferior quality or it was removed some time before it was worn out. For general estimates the averages and totals may be considered to closely represent a fair average statement of the cost and tonnage.

In the second table the cost at shafts per lift of one hundred feet is shown to average .053 cents per ton. This is equivalent to 0.38 cents per ton of material raised, closely approximating the figures (0.36) given by Taylor, above cited.

Comparing the cost in shafts and slopes we find apparently .069 and .053 cents per ton for each one hundred feet of lift;—but it must be remembered that the slope lifts are *slope* measurements of *length*, not of vertical depth, therefore the actual cost for each one hundred feet of *vertical* lift in a slope will be much greater than 0.069 cents, as given in the table. If the slope dips 30° a slope length of 200

Table showing the Cost of Wire Ropes per ton of coal.

SLOPES.	Number of Ropes.	Diameter in Inches.	Length in Feet.	Cost.	Tons raised by each rope.	Total tons of Coal raised.	Cost per ton in cents.	Cost per ton per lift of 100 feet.
1, ...	6	1¾	900	$3,240	66,616	375,700	0.86	0.095
2, ...	2	2	1,000	1,520	98,280	196,560	0.77	0.077
3, ...	1	1¾	850	510	203,700	203,700	0.25	0.029
4, ...	2	1¾	1,000	1,200	37,175	74,350	1.61	0.161
5, ...	1	1½	1,200	540	37,500	37,500	1.47	0.122
6, ...	1	1¾	1,100	660	77,700	77,700	0 85	0.077
7, ...	2	1⅜	950	760	41,825	83,650	0.90	0.094
8, ...	1	1½	950	428	70,950	70,950	0.60	0.063
9, ...	1	1½	675	304	102,200	102,200	0.29	0.043
10, ...	4	1¾	820	2,952	149,037	596,150	0.50	0.061
11, ...	2	2	1,050	1,596	166,650	333,300	0.48	0.046
Averages and totals, . . .	23		933	$13,710	93,555	2,151,760	0.64	0.069
SHAFTS								
1, ...	2	1¾	925	$1,100	88,725	177,450	0.63	0 068
2, ...	2	1¾	635	762	117,180	234,360	0.32	0 051
3, ...	2	1¼	500	350	86,222	172,445	0.20	0 041
Averages and totals, ...	6	. . . .	687	$2,322	97,376	584,255	0.38	0.053

feet will represent a vertical lift of 100 feet and the cost will therefore be $0.069 \times 2 = 0.138$ for a lift that in a shaft will cost but 0.053 cents.

These tables have been calculated from the actual tonnage raised by each rope. As the coal raised does not exceed two-thirds this amount, the cost *per ton of coal* should apparently be correspondingly increased, but the value of the discarded rope, which is probably worth one-third its first cost, is a nearly equal off-set to this, and the figures above given may therefore be considered to represent the cost *per ton of merchantable coal.*

Assuming the above average figures of cost per ton at slope and shaft collieries to represent the *ratio of wear*—*i. e.* 138 : 53—we have a means of approximately determining the relative amount of wear due to sheaves and drums and to knuckle and bearing pulleys (plus friction caused by dragging on the ground).

Assuming an equal wear per ton per vertical lift of one hundred feet at both shaft and slope collieries from the friction due to drums and sheaves (.053), we have 138—53 =85 representing the wear from friction on the knuckle and bearing pulleys; or a ratio of 85 : 53 as the ratio of the wear on the slope to that caused by drums and sheaves, being 61 and 39 *per cent.* respectively.

The wear on bearing pulleys in a slope is from a rolling friction, and can be lessened only by proper attention to the condition of the pulleys and a free use of lubricants; but the wear occasioned by knuckle and deflective pulleys partakes of the character of the wear on sheaves.

The policy of winding wire rope on drums of comparatively large diameter has been so frequently and urgently advocated by wire-rope makers, that mine superintendents now very generally use drums larger than the minimum diameters advised by the manufacturers. This general increase in size is doubtless due in part to the growing necessity of increased rapidity in winding.

The John A. Roebling's Sons company publishes a circular, in which the minimum drum diameters are as shown in the first column of the following table; the second column is taken from the Hazard Manufacturing company's circular; in the third column are figures given by a rule largely used by mine superintendents, as follows: "For every quarter of an inch in the diameter of rope allow one foot in the diameter of the drum." This rule creates a constant ratio of 1 : 48 between the diameters of rope and drum.

ROPE DIAMETER IN INCHES.	DRUM DIAMETER IN FEET.		
	Roebling's.	*Hazard.*	*Common Rule.*
$\frac{1}{4}$	....	....	1
$\frac{1}{2}$	$1\frac{1}{2}$	2	2
$\frac{5}{8}$	2	3	$2\frac{1}{2}$
$\frac{3}{4}$	$2\frac{1}{2}$	$3\frac{1}{2}$	3
$\frac{7}{8}$	$2\frac{3}{4}$	$3\frac{3}{4}$	$3\frac{1}{2}$
1	3	4	4
$1\frac{1}{8}$	$3\frac{1}{2}$	$4\frac{1}{2}$	$4\frac{1}{2}$
$1\frac{1}{4}$	4	5	5
$1\frac{3}{8}$	....	$5\frac{1}{4}$	$5\frac{1}{2}$
$1\frac{1}{2}$	$4\frac{1}{2}$	$5\frac{1}{2}$	6
$1\frac{5}{8}$	5	6	$6\frac{1}{2}$
$1\frac{3}{4}$	$6\frac{1}{2}$	$7\frac{1}{2}$	7
$1\frac{7}{8}$	....	$7\frac{3}{4}$	$7\frac{1}{2}$
2	7	8	8
$2\frac{1}{4}$	8	$9\frac{1}{2}$	9

The figures of the above table are necessarily arbitrary,—determined not by theory, not by experimentation, but by the "rule of thumb" process,—but while necessarily imperfect, they are probably as good for practical use as anything we shall ever obtain.

Theoretically, to insure the minimum wear the drum diameter should be infinite, *i. e.*, the rope should not be bent at all. But as bending is an unavoidable concomitant of winding, it is of great importance to arrange the winding machinery so that the rope shall suffer the least possible deflection from a straight line of any given length; or in other words, the radius of curvature in every case should be as great as circumstances permit. The table is given to indicate the *least diameter* of drum upon which ropes of certain sizes should ever be bent, no matter what the circumstances that seem to require an infringement of the rule,—they are *not* intended as dimensions for ordinary use.

A certain amount of external friction upon the drums and sheaves, and in slopes upon the pulleys, is always unavoidable and need receive no attention other than an endeavor to reduce it to a minimum by proper sheave linings (?) and lubricants.

But in shafts the wear from this cause is insignificant compared to the great internal wear caused by the rubbing and grinding of the wires of each strand upon those in contact with it, and of the wires of each strand upon each other, whenever the rope is forcibly flexed or bent and simultaneously subjected to great tensional strain.

When an elastic bar of iron or a wooden plank is bent, the under (concave) fibers suffer a strain of compression and the upper fibers one of extension; but as in a wire rope the strands cross diagonally from top to bottom, each strand accommodates itself to the new condition by *slightly moving* upon those in contact with it. The weight of the load is thus equally distributed.*

The sand and dirt which inevitably finds its way into the body of a rope must always largely accelerate the rate of wear.

The following propositions are almost self-evident:

1. The internal wear is proportional to the friction;
2. The friction is proportional to the *amount of movement* between the strands and wires when the rope *adjusts itself to a curved surface, or readjusts* itself to a straight line;
3. The amount of this movement depends directly upon the *degree of curvature*, which is *inversely proportional* to the diameter of drum or sheaves.

Hence: The internal wear from friction is inversely proportional to the diameter of drum or sheave.

But while mine superintendents very generally recognise these principles in determining the size of drum best adapted to their purposes, they have not, as a rule, in the past placed nearly as much importance upon the diameter of the sheaves and pulleys over which the rope runs. Propositions two and three show that the wear occurs *at the moment the rope is flexed in passing upon the drum*, hence precisely the same wear must be occasioned by flexing the rope on a sheave of equal diameter ; *and mark*, as the rope almost immediately passes off the sheave on its route to the drum and must *readjust itself to a straight line*, (see prop. 2 above) the wear caused by the sheave is precisely *double* that caused by a drum of equal size. Yet we often see a $1\frac{3}{4}''$ rope bent over *sheaves* six feet or less in diameter and wound on *drums* twice that size. As regards the internal frictional wear, it seems far better to decrease the size of drum than to diminish the diameter of the sheaves. In the Roebling's Sons' company's circular, the minimum figures are expressly stated as being for *drums* or *sheaves ;* and these figures are the minimum diameters that can prudently be employed ; larger sizes give better results.

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* If the strands were simply bound together and not twisted in a helix, the upper strands would sustain all the weight, the lower ones in contact with the sheave being subjected to a strain of compression.

I approach the subject of knuckle pulleys and deflection pulleys with much hesitation. The wear caused by a knuckle pulley used to effect a slight change in the direction of a rope, is influenced by precisely the same conditions that affect the wear on sheaves,—provided, always, that the angle of deflection is sufficient to cause the rope to *conform to the curve of the pulley* for an appreciable distance along its lower or contact surface. This will vary with the size and stiffness of the rope and the working load.

After much fruitless labor, I am convinced that it is not possible to deduce a law or express by a formula, the exact diameters of deflection pulleys which shall cause no greater wear than the minimum size of drum, for rope of a given thickness ; but it seems patent that in any case where the deflection is 15° or more, the knuckle or deflection pulley becomes to all intents and purposes a sheave, and should be treated as such.

For deflections of less than 15° and more than 5° it might be well to use pulleys of size proportional to the deflection, but for any deflection exceeding 15° the pulley should certainly be equal in diameter to the minimum size of drum or sheave for rope of equal thickness.

Wire ropes used for hoisting generally contain 7 strands of 19 wires each; or six strands of 19 wires wrapped round a hemp-rope center. Hemp-center ropes generally wear better than those with wire centers, and are preferred in the Anthracite districts.

The following table shows the solid metal cross-section, in square inches, of ropes of 19 wires to the strand, for wire and hemp centers, from which the breaking weight can be calculated by multiplying the cross-sectional area by the strength of material (steel or iron) per square inch. The working load should not exceed one-fifth of the breaking strain. The table also shows the approximate weight per foot of wire and hemp center ropes, and Roebling's figures for breaking strain, working load, and price list.

DIAMETER.	AREA OF METAL.		WEIGHT PER FOOT.		BREAKING STRAIN.	WORKING LOAD.	PRICE.
Inches.	Wire Center. Inches.	Hemp Center. Inches.	Wire Center. ℔s.	Hemp Center. ℔s.	Tons of 2,000 ℔s.	Tons of 2,000 ℔s.	Hemp Center. Cents.*
$\frac{3}{4}$	.36- .30	.31- .26	0.92	0.83	8.64	1.75	15
$\frac{7}{8}$	.50- .45	.41- .38	1.25	1.14	11.50	2.5	19
1	.65- .60	.57- .50	1.65	1.50	16.	3.	22
$1\frac{1}{8}$	.82- .75	.71- .62	2.09	1.90	20.	4.	27
$1\frac{1}{4}$	1.0 - .90	.83- .78	2.59	2.33	27.	5.5	33
$1\frac{3}{8}$	1.2 -1.1	1.10-1.0	3.27	2.94	. . . . .	. . . . .	. . . .
$1\frac{1}{2}$	1.4 -1.3	1.3 -1.1	3.71	3.10	35.	7.	41
$1\frac{5}{8}$	1.7 -1.5	1.5 -1.3	4.38	4.00	44.	9.	50
$1\frac{3}{4}$	2.0 -1.8	1.8 -1.6	5.06	4.55	54.	11.	60
$1\frac{7}{8}$	2.3 -2.1	2.0 -1.8	5.80	5.22	. . . . .	. . . . .	. . . .
2	2.6 -2.4	2.3 -2.0	6.61	6.02	65.	13.	76
$2\frac{1}{4}$	3.3 -3.0	2.9 -2.5	8.36	7.52	74.	15.	98

The breaking strain in this table is evidently underestimated; for 65 tons of 2,000 ℔s.=130,000 ℔s., which divided by the metal cross-section of a two-inch hemp-center rope (2.2), gives only 59,090 ℔s. strain per square inch of metal cross-section, which is a low estimate for good wire metal.

Some other points referred to in the above-mentioned circular are of value, especially the instructions relative to pulley linings.*

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* Cost of wire-center ropes is about 10 per cent. more than hemp-center ropes. Date of price-list Oct., 1880.

It recommends that all sheaves and pulleys should be lined with blocks of hard wood set on end, and renewed as fast as they are worn out. "When large sheaves run with great velocity the grooves should be lined with leather, set on end, or with India rubber. This is done in the case of all sheaves used in the *transmission* of *power* . . which frequently run at the rate of 4000† feet per minute."

The advantage of steel ropes over wire ropes seems to be confined to their relatively slower rate of wear. They are somewhat, but not very much, stronger than iron-wire ropes.

Coal tar, coal oil residuum, heavy lubricating oil or any other lubricant that is water-proof and will adhere to the rope, may be used as a preservative. Raw linseed oil or oil mixed with equal parts of Spanish Brown or lamp-black, is also recommended. For rope used underground, or in water, a mixture of one barrel of tar and one bushel of fresh-slacked lime may be used,—applying it hot to the rope.

Galvanizing rope is useless, as the zinc is immediately worn off and the rope then rusts very rapidly.

Wire rope should never be *coiled* like hemp rope, as the twisting will displace the strands from their proper relative position. It must always be rolled or unrolled from the bobbin or reel as from a drum.

On slopes eight or nine hundred feet deep the ropes are commonly replaced at intervals varying from six months to two years.

After a rope has been spliced, turned end for end, shortened by cutting off bad ends, and is no longer safe to use on a main hoisting slope or shaft, the customary practice is to use it on an underground (inside) plane, a dirt plane, or in the breaker, where shorter lengths are needed and the strain is not so great. A rope is thus sometimes used on three or four different planes before it is thoroughly worn out.

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*It is questionable whether any benefit is derived from sheave linings for large sheaves used in ordinary colliery practice. Some engineers are opposed to linings for head-sheaves, preferring a perfectly true groove in the rim made to fit the rope accurately.

† If my memory serves me, power is transmitted by wire rope running 160 feet per second, or 9600 feet per minute, (a speed of 100 miles an hour,) at the Calumet and Hecla mines on Lake Superior.

Partly worn ropes are frequently sold for use at slate and stone quarries.

The sockets are put on by spreading the wires in the socket, drawing them all taut and pouring the socket full of lead. It is well to surround the socket and rope with wet clay to prevent the wires outside the socket from becoming too hot, as this will impair the strength of the rope.

Flat ropes.—Although many advantages have been claimed for flat rope, it seems that the only well-founded claim is that of the counterbalancing action of the rope coiling upon itself. Flat ropes are still used at many mines in the west, but in the anthracite coal regions they are now almost unknown.

The advantage gained is more than counterbalanced by the increased wear from the rope lapping upon itself, and as the same counterpoising can be effected in several other ways, the use of flat ropes, in any but very exceptional instances, does not seem advisable.

The plan of winding a round wire rope upon itself on a narrow drum, confined between two metallic discs, has been successfully used at the Wartet colliery in France, and in England at Radcliff, Lancashire. It is questionable whether this plan is any better with round than with flat ropes.

The conical drum with a thread or groove (scroll) for the reception of the rope is undoubtedly one of the best means of equalizing the load.

Tapering ropes are not used in the Pennsylvania coal fields. When the deepest parts of our coal basins are penetrated by shafts from 1600 to 2400 feet deep, it may be advantageous to introduce ropes of unequal section, but for the comparative shallow shafts of to-day they are not necessary. They are not at all adapted to deep slopes, on which the greatest wear always occurs at the *lower end* of the rope, as that portion travels further and is subjected to greater friction than any other part.

CHAPTER XVI.

Safety attachments, Signalling apparatus, and Indicators.

In this chapter all of the additions to a colliery plant especially designed to reduce the probabilty of accident to a minimum, or to decrease the risk, both to life and property, from the occurrence of breakage, or from carelessness, will be taken up,—in the following order:

1. *Safety catches* on shaft cages, to prevent the cage from falling if the rope breaks;

2. *Bridle chains* to hold the cage (if the rope breaks at the rope socket) if the draw-bar breaks;

3. *Bonnets* to protect the cage, car, or miners from objects falling down the shaft;

4. *Detaching hooks* and other appliances to prevent overwinding;

5. *Automatic governors* and *steam brakes*, as a means of stopping a runaway engine;

6. *Slope cage catches;*

7. *Drags* on slope carriages, barneys, or mine cars, to stop the cars in case the rope breaks;

8. *Gates and fences* to protect the mouths of shafts and prevent persons from falling into them;

9. *Signal wires*, *speaking tubes*, and other means of communication between the miners below and those in charge at the surface;

10. *Indicators* to show the position of the cage or cars in the shaft or slope and other methods of determining this.

Safety catches.

The law requires "an improved safety catch . . . on every carriage used for lowering or hoisting persons" . .

but the context does not make it perfectly clear that this provision is intended to apply to cages used on slopes.

"*An improved safety catch*" is a rather vague term, but not more vague than many other phrases occurring throughout the act and its amendments. As the meaning to be given the word "improved" is not specified, it becomes practically a dead-letter, and the requirement to be fulfilled is simply that a safety catch be placed on each cage used for raising or lowering persons.

A great many different forms of safety catches are in use, and most of them are efficient appliances.

The report of the "Committee of Safety cages," appointed by the North of England Institute of Mining Engineers (Transactions Vol. XIX, 1869-70), shows that from 1857 to 1868, inclusive, the number of deaths from broken ropes or chains was 185, and the coal mined in that period 1,043,000,000 tons, or one death to about five and one half million tons of coal mined. If the same ratio obtained in this country the number of deaths (safety catches not used) would be about five per year.

It also shows that a man might ride on a cage 400 times a year for 12,500 years before meeting his death in this way, stating that while "it cannot be denied that ascending and descending into a pit is in itself a dangerous occupation . . . it is found that by careful selection of ropes, chains, and other tackle, and a watchful and close inspection of the winding apparatus and shafts, the risk is so reduced that it hardly approaches that of a walk for an hour in the streets of London. *It must be admitted that the safety apparatus that succeeds in reducing this risk must combine elements of no common excellence.*"

That the adoption of safety catches introduces an additional source of accidents, and that it increases the number of casualties by relaxing the vigilance of those in charge of the winding machinery, is doubtless true; but while the number of rope breakages is increased, the number of deaths from this cause should be decreased, if the safety catches adopted are really efficient devices.

Three kinds of springs are used to throw the catches against the guides; 1. Rubber cushions, or spiral springs surrounding the draw-bar, often confined within a cylinder; 2. Straight or curved steel springs; 3. Spiral springs surrounding the shafts which carry the dogs.

Chisel-pointed bars thrown out against the sides of the guides have been found uncertain in action, and split and otherwise damage the guides; clamps which grasp the guides and hold the cage by friction only, have not given good results; cam-shaped dogs with a chisel edge have been found in practice to be unreliable, and often do considerable damage to the guides; the catches giving the best results and those which are used to the almost entire exclusion of other forms are cam-shaped dogs with a serrated surface that grasps the guide. Catches of this description are dependent upon the spring only to throw them against the guides with sufficient pressure to make them bite,—the weight of the cage then forces them firmly upon the guides with a pressure that theoretically approaches the infinite.

Rubber cushions surrounding the draw-bar are now largely used in place of other springs. Straight or bowed springs are more likely to become deranged than other forms and are more difficult to adjust to the required tension.

The safety catches at the Pottsville shaft are operated by spiral springs surrounding the rods or shafts which carry the dogs.

Atlas Sheet No. XI shows different forms of safety catches in use in the Wyoming district. Some engineers prefer placing the dogs nearly opposite the cage platform, others place them near the top of the cage. The latter arrangement seems preferable because it simplifies the construction of the catch.

One of the chief difficulties that has been experienced in the use of safety catches is their tendency to be thrown into action by any sharp jars or shocks to which the cages or winding machinery may be subjected, by suddenly stopping the engine, or when the cage is landed at the top or bottom; but accidents from this cause are now extremely rare, and if proper care is taken in adjusting the springs and catches, there seems to be no reason why any such difficulty should occur.

Practical tests of the catches in use, made by hanging the cage and allowing it to drop, show that they are, as a rule, very efficient devices. The dogs usually take hold at once, the cage only dropping a few inches, or at most a few feet. When the guides are very greasy or wet, the trial tests are sometimes not so satisfactory, and the cage may drop several feet before the dogs take a firm hold and stop it, and the results are still less satisfactory when the guides are covered with ice; but even in this latter case the cage sometimes drops less than one foot.

Fortunately for the utility of safety catches, ropes are usually broken while a loaded cage is being raised, and the cage has an *upward* momentum; if a rope breaks on the empty side and when the cage is rapidly descending (at a speed say of thirty or forty feet a second) its momentum is so great that either the catches must break, or the cage or guides and shaft lining will be torn to pieces,—the catches generally hold, and either the guides or cage suffer more or less injury under such circumstances.

In experimental tests with ice-covered guides the cage has been known to fall eight to fifteen feet before the dogs ploughed their way through the ice and took firm hold of the guides, but the momentum the cage acquired was so great that the guides were destroyed.

Bridle chains.

The law provides (*Sec. 10*) that "the main link attached to the swivel of the wire, or any other rope, shall be made of the best quality of iron and tested by weights, or otherwise satisfactorily to the inspector, and bridle chains shall be attached to the main link from the cross pieces of the carriage, and no single link chain shall be used for lowering or raising persons into or out of any said mines . . . ."

I have not been able to satisfy myself of the exact import of this clause; the "main link" is by some interpreted to mean the rope socket and the bridle chains are accordingly arranged as shown on the carriage designed by Mr. F. B. Parrish (Atlas Plate No. IX.)

When this plan is adopted,—and if the law be construed literally as worded, it or a worse arrangement, that of attaching the bridle chains to a chain link connecting the swivel (when used) to the rope socket, must be adopted,—the bridle chain offers no protection in case the rope breaks at or near the rope socket.

Some engineers attach the bridle chains to a clamp made in two pieces, which when bolted together firmly clamp the rope. This clamp is placed from three to five feet above the rope socket, and when properly constructed and carefully adjusted is doubtless a much better method of fastening the bridle chains; but if improperly constructed or adjusted it may injure and weaken the rope.

Bonnets.

The law provides for "a sufficient cover over-head on every carriage used for lowering or hoisting person," but the covers provided are not always "sufficient." They are frequently broken, or very much weakened by rust, and are not renewed as frequently as could be desired.

They are usually inclined, so that objects falling upon them will slide down and drop upon the cage, and to prevent objects of moderate size from wedging between the edge of the cover and the shaft lining, they are usually made shorter than the cage, so that a space of one foot or more is left between the lower edge of the bonnet and the shaft lining or buntons.

Boiler iron plate is now used to the almost entire exclusion of other materials for these cage covers, but it is an open question as to whether a covering of plank would not give as good results, longer wear, and be more easily replaced.

Detaching hooks.

Appliances for the prevention of overwinding are considered unnecessary and even mischievous by nearly all anthracite mining engineers.

Dependence must be placed upon the engineer in charge of the winding engine, upon sufficient head-room, upon the efficiency of the brake, and the engine itself, for the prevention of accidents of this class,—such is the opinion of a large majority of mining engineers and colliery superintendents; an opinion probably justified by the imperfections of the detaching hooks and other appliances that have been tried in other districts, and the dangerous feeling of security following the adoption of devices of this class.

A mechanical engineer of large practical experience in the building and operating of winding engines once remarked in my hearing that the only successful device to prevent overwinding would be one that threatened the life of the engineer in charge; as he expressed it, "a ten-ton weight suspended over the engineer so that it would fall whenever the cage passed a certain point, might decrease the number of over-winding accidents, but the adoption of detaching hooks and other appliances will increase the number of casualties."

The difficulty with nearly all of the detaching hooks that have been invented is, that, while they work successfully in trial tests when the parts are in perfect order, properly lubricated, and free from rust, they may become rusted (or frozen) and instead of unlocking and detaching the rope, the rope is broken and the cage falls or is caught by the safety catches; and in addition to this objection they introduce a new element of danger, viz: the possible accidental detaching of the rope while the cage is in the shaft or at the top or bottom landing.

A thoroughly reliable detaching hook, open to none of these objections, if used on a plant in charge of an able engineer having *no faith* in the appliance, would certainly constitute a valuable addition to the winding plant; but it is doubtful whether any hook yet devised is reliable at all speeds of winding and in any condition, and it is almost impossible to find an engineer who would not place too much reliance upon it.

The Pottsville plant (Page plate No. 33) is provided with detaching hooks. The detaching ring placed on top of the guide frame, and even the hook (on the right hand rope) itself are shown by this small photographic view of the head-frame. I do not know that any other plant in the anthracite regions is provided with detaching hooks.

This plant is also provided with another safeguard against overwinding. When the indicater arm moves a certain distance beyond the landing mark, the steam brake is automatically applied and the main throttle valve closed.

The present feeling is strongly against the adoption of all devices intended (by automatic action) either to render overwinding impossible, or to decrease the risk by detaching the cage from the rope; and this feeling is grounded upon the following objections to devices of this class:

1. They inspire the engineer with a misleading feeling of security;
2. They are more or less complicated in construction, require frequent lubrication and inspection, and destroy the simplicity of the plant;
3. They may be the direct cause of accident by introducing new elements of danger;
4. Their cost;
5. They are not thoroughly reliable.

The safeguards against overwinding upon which nearly all engineers and superintendents now rely may be briefly stated:

1. The employment of a sober, reliable, and competent engineer, to be held personally responsible for overwinding accidents;
2. A good brake, and a winding engine thoroughly under the control of the engineer;
3. A reliable method of indicating the position of the cage, whether by indicator, by mark on the rope, or by both;
4. Sufficient height of sheaves to allow about two-thirds of a revolution (or more) of the drum between the landing and the sheaves.

Steam brakes and "Governors."

The arrangement of steam brake and governor by which the brake is automatically applied whenever the speed of winding exceeds a certain maximum, has, to my knowledge, been adopted only at the Pottsville shaft.

Probably the best arrangement is to make the steam cylinder act as an auxiliary to the hand lever, so that the latter may be successfully used when steam is shut off. At some English collieries the breaks have been operated by a weight, and the steam used only to hold the brake away from the drum, but this plan has not been adopted here, and is open to several objections.

The great difficulty with the hand brake is to multiply the pressure sufficiently and at the same time obtain sufficient movement of the brake band to throw it entirely off the brake ring. This may be obtained by abandoning the ordinary iron brake band for a brake block, and there are few reasons why this change should not be made. As the friction is independent of the surface, but varies directly with the pressure applied, it is evident that a properly constructed brake block will offer as much resistance as a band completely encircling the brake ring, and a block may be adjusted with very small clearance so that a movement of one-eighth of an inch will throw it off. If the hand lever moves through an arc of thirty inches, the pressure applied by the engine will be increased two hundred and forty times, thus a push of fifty pounds against the lever creates a pressure of twelve thousand pounds on the drum.

By locating the brake block beneath the drum the only effect is to take some of the weight of the drum off of the journals and pedestals. To attain the best results with a brake of this form it will be necessary to use materials that give a high co-efficient of friction.

Brakes of this form are coming into use at some very large English collieries, and we shall doubtless soon see them tried in this country on some large winding plants

They require a very much stronger brake ring than the ordinary band brake.

Slope cage catches and Drags.

These are needed only on very steep slopes and may be made similar in every way to the catches used on shaft carriages, but as the law does not especially provide for their addition to cages used in slopes, their use has seldom been suggested.

Slope cages used on moderate and even on very steep dipping slopes, are occasionally provided with a "drag" or with bars * that drop when the rope becomes slack or breaks.

These catches or drags act exactly like the drag on a heavy teaming wagon which holds the wagon and prevents it from pulling down hill while the horses are resting, or in case the harness breaks, but as the slope is usually much steeper than a road, the drags may lift the cage, barney, or car off the track and pin it against the roof of the slope.

The use of the drag is as yet confined principally to mine cars, and to a small number of collieries.

The objections advanced by those opposed to this form of safety catch, are:

1st. They are useless on the empty trip,

2nd. They may become unhooked and act on the empty trip and throw the cars off the track,

3d. They are useless on very high slopes,

4th. They are almost useless on self-acting planes, as they can only be used on the empty trip.

If properly constructed, there should be no danger of the drag becoming accidentally detached from its fastenings and acting on the empty trip; the number of slopes so high that the cars can rise and fall over the drags is comparatively small; and as they are seldom needed on the empty trip,—the rope generally breaking on the loaded trip,—these objections to their use do not seem to be well founded. The amount of money and the time saved by a successful drag in preventing the destruction of mine cars, and the protection they afford the bottom men, are advantages not to be lightly disregarded.

One of the most simple drags in use consists of an iron bar about four feet long attached to the hind axle of the car. When not in use it is suspended in a hook; when in the slope it is allowed to trail on the floor.

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* Improperly called a drag.

A drag of this form has been in use at the Mount Pleasant slope at Hyde Park for twenty-three years and has given entire satisfaction.

A single drag is sometimes attached to the hind car of each loaded trip. On slopes of moderate pitch a single drag may be all that is necessary to hold three or four cars, but on steep dips it is evident that greater safety is assured by the use of a drag on each car. The advantage of using a detachable drag of this kind is evident; it is attached to the coupling of the hind car of the loaded trip by the bottom-men, when the trip arrives at the top it is detached and sent down in an empty car,—the risk of accident to the empty trip is thus avoided.

Gates and fences.

The law provides that "the top of each shaft shall be securely fenced by vertical or flat gates, covering the area of said shaft, and the entrance of every abandoned slope, and air or other shaft, shall be securely fenced off." While the wording of this clause is rather ambiguous its import is clear. It is evidently intended to prevent persons falling down shafts and slopes.

The number of such accidents is not small, but it has doubtless been decreased by the provisions of this act of the Legislature.

The plan generally adopted is to fence in all portions of the shaft mouth except that part opposite the winding compartments and to close these with gates. On the side the cars run off the carriage, the gates are frequently made double and hung so that they will close without assistance. On the other side a heavy beam forms the gate to prevent empty cages running into the shaft. This beam is lifted and held out of the way by the cage, so that the car can pass beneath it.

Another plan is to place vertical sliding gates at all the openings; these are lifted by the cage and settle into place when the cage descends.

All of these are satisfactory methods. Flat (horizontal) doors are not used except to close air-shafts which may be used in emergency for raising the miners, or occasionally for lowering mine supplies, etc.

The law does not require gates at landings below the surface, but in shafts and in steep pitching slopes they are frequently used, not so much as a safe-guard against accidents to miners, as to prevent cars and other objects from falling down the shaft or slope.

It is doubtful whether the adoption of gates decreases the number of accidents from miners and laborers falling down shafts and slopes, but it certainly does reduce the risk of accident from cars and other objects falling down.

Signal wires and Speaking tubes.

The law of 1870 requires (Sec. 10) every colliery operator to "provide and maintain a metal tube from the top to the bottom of such slope or shaft, suitably calculated and adapted to the free passage of sound therein, through which conversation may be held by and between persons at the bottom and top of the shaft or slope, and also the ordinary means of signalling from and to the top of the shaft from the bottom, ..."

The "ordinary means of signalling" is by a wire running from the bottom to a gong, or hammer and plate, in the engine room, and is for the purpose of signalling to the engineer when everything is in readiness for him to hoist the cage, of informing him whether he is raising coal, rock, or men, and where he is to stop the cage.

Two signalling wires are generally used, one running as above described and the other running from the mouth of the shaft or slope to the engine room. The engineer does not start the engine until he has received two signals, one from the top and one from the bottom, indicating that everything is ready for another winding.

The signals vary at different collieries ; they generally run from one to four or five strokes ; two strokes from the bottom is commonly used to indicate that everything is ready, three strokes that men are on the cage or car to be raised, etc., but there is no uniform system.

When coal is raised from several different levels, additional wires with gongs of varying pitch, (or hammer and plate signals) are added, but two landings are often worked with only one signal wire.

A rough sketch of a "hammer and plate" (a piece of boiler plate iron) signal is shown by Fig. 49. This method of signalling is preferred by many to the use of gongs.

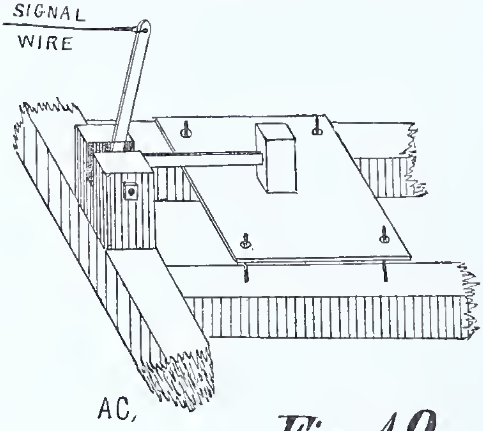

Fig. 49.

Speaking tubes are generally made of two-inch iron tubing. In some cases the tube is run into the engine room, but often extends no further than the mouth of the shaft or slope. It is seldom provided with a whistle, as in a tube of this size and with a length of three to twelve hundred feet it is difficult if not impossible to blow a whistle. When the top-man wishes to communicate with the bottom (and *vice versa*) he simply raps on the tube with an iron bar, a stone, or any hard substance, and the noise thus made is perfectly audible at the other end of the tube.

An ingenious device has lately been added to many speaking tubes, especially at shaft collieries in the Wyoming district, whereby the speaking tube is converted into a signalling apparatus. Small brass cylinders about six or eight inches in diameter by about six inch stroke are attached to the tube. These cylinders form small air pumps ; when the piston is thrust in by a quick push on a knob affixed to the end of the piston rod, the compressed air operates the clapper of a gong situated at the other end of the tube.

While this device requires the occasional inspection of a skilled mechanic to keep it in perfect working order, it is probably less likely to cause as many and as frequent stoppages as those occasioned by the breakage of signal wires.

Two or more speaking tubes are frequently used for communication with different levels, but branch tubes are sometimes used.

The wire used for signalling is rarely smaller than ordinary telegraph wire (No. 8,) and often much thicker,—from No. 6 to No. 2. Rapid oxidation from mine water, moisture in the air, and the sudden jerks to which it is subjected make large sizes necessary. Small wire rope has been used, but unless kept thoroughly greased it rusts very quickly, and is objectionable on account of stretching, and of its tendency to kink unless kept stretched tight.

Indicators.

The indicator attached to the Pottsville winding plant has already been described. Indicators of a much more simple type are commonly used.

A very common method is to insert an iron pin, about an inch in diameter, into the center of the drum shaft, and wind a small chain or cord around the pin. This cord is carried over a small sheave or pulley, and a weight carrying a pointer, a block of wood, a ball, or a slide carrying a gong, is attached to the other end. The weight or pointer rises or falls in front of a board, or between guides, on which lines are marked to indicate different points in the shaft or slope. When a gong is used, adjustible striking points are fastened to the guides by set screws, as shown by Fig. 48.

Some indicators are made so that the chain or cord *entirely unwinds* and is *again wound up* during each winding.

The same object may be accomplished by using two indicators—one for each cage—but this plan is open to many objections.

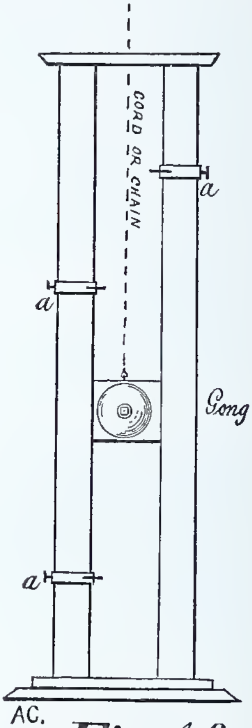

Fig. 48.

The addition of the gong to indicators of this class is undoubtedly advantageous. It constitutes an additional safeguard against overwinding, and is open to no well-founded objection.

Indicators operated by a cord or chain wound around a pin on the drum shaft, or around some revolving portion of the winding gear, are more common than any other form. It is, however, evident that they are not entirely reliable, but if the cord or chain overlaps in winding, the error of the indicator is always on the safe side, for it indicates the arrival of the cage at the top *before* it reaches the landing; but if during the *unwinding* of the cord or chain the latter becomes caught or entangled with coils still on the shaft, or if it binds in the sheave or pulley through which it runs, the cage may reach the landing before the position of the pointer indicates its arrival.

Atlas Sheet No. VII shows the indicator used at the Laurel Hill colliery. The pointer travels back and forth in front of a black-board, on which the different landings are indicated by white paint or chalk marks. The pointer is moved by the rotation of the screw shaft, which is driven by motion taken from a drag crank and transmitted by two pairs of bevel gears.

When the drum in not geared direct to the engine, the indicator is frequently driven by a bevel gear from the drum shaft or a bevel gear bolted to the drum spiders.

The Laurel Hill indicator is also made to do duty in scoring the number of cars raised. At each winding the pointer catches in a ratchet wheel behind the two dial faces shown in the drawing, and the number of cars is shown at any time by reference to these dials. This device is intended to replace, or act as a check upon, the car-board. on which the number of cars raised is usually scored by wooden pegs.

Indicators of this type, driven by positive gearing from the winding machinery, are the only ones regarded with confidence. Chain or cord indicators are distrusted by nearly all engineers, but are used by many partly because they are cheap, and partly because the engineers in charge of winding engines very generally depend upon a mark on the rope and not upon the indicator.

If indicators are used they should be trustworthy, and not liable to derangement from trivial causes. If this be admitted, then all chain and cord indicators should be abolished and replaced by indicators with positive motion, driven by gearing from the winding machinery.

But the engineer in charge of the winding engine rarely depends upon the indicator for a knowledge of the exact position of the cage or cars at the top of a shaft or slope; he almost invariably relies upon white marks made on the last coil or coils of the rope,—this plan he considers more accurate, more trustworthy, and less likely to lead him into error, than any indicator a machinist can construct.

This faith in the rope as an indicator has spread to many mining engineers and superintendents, so that we now find many large collieries not provided with indicators.

The greatest fault of nearly all indicators is that they give a regular movement throughout the winding, and the space over which the pointer travels for the last (or any) revolution is too small to enable the engine-man to land the cage or cars accurately.

The indicator at the Pottsville shaft is not open to this objection, but its complicated construction, the necessity of reconstructing it to adapt it to winding from a greater depth, and its cost, will probably prevent its general introduction.

Indicators of the type shown by Atlas Sheet No. VII are provided more frequently with dial disks and revolving pointers than with the straight back and forth movement of the Laurel Hill indicator.

While it is probably true that the use of an indicator of ordinary construction constitutes a safeguard against overwinding and other accidents in no way superior to that furnished by marks on the rope, the indicator is an undoubted aid when there are several landings in a shaft or slope, and especially when some of these are close together.

There can be little doubt but that an engineer relying upon the rope will handle his engine more cautiously, and watch the mark more carefully, than an engine-man running by an indicator, especially when the indicator is furnished with a gong.

The marks referred to are sometimes made by simply daubing some white paint on the rope and drum in such a position that when the landing is made the marks will be in line with some prominent, fixed object in the engine house,—often a part of the machinery,—they are sometimes made by wrapping cord or tarred rope around the rope at certain points, but as the cord may slip this is not a good plan.

As the marks on the dial, horizontal black-board, or upright guides, to indicate points in the shaft or slope must be changed from time to time on account of the rope stretching, etc., they are frequently made with chalk. As these chalk marks may be tampered with by malicious persons, (and for other obvious reasons,) this is not a safe method of indicating the position of the cage, and it constitutes another objection to the use of indicators.

Some mining engineers insist upon the use of indicators on all winding plants, others are strenuously opposed to them under any conditions. I think both these plans are suited to certain cases, and that it will be found best to use indicators of approved forms with gongs attached at some collieries, and to dispense with them altogether at other workings.

CHAPTER XVII.

Access to and from Mine Workings.

The following extract from Section 3 of the Act of March 3d, 1870, will be more clearly understood by reference to the foot notes which I have introduced to indicate the probable import of several rather obscure words and passages:

. . . "it shall not be lawful for the owner or agent* of any anthracite coal mine or colliery, worked by or through a shaft or slope, to employ any person in working† within such mine or colliery or to permit any person to be in such coal mine or colliery for the purpose of working† therein, unless there are (,?) in communication with every seam or stratum of coal worked in such coal mine or colliery, for the time being at work, at least two shafts, or slopes, or outlets separated by natural strata, of not less than one hundred and fifty feet in breadth, by which shafts, slopes or outlets distinct means of ingress and egress are always available to the persons employed in the coal mine or colliery; but it shall not be necessary for two shafts, slopes, or outlets to belong to the same coal mine or colliery; if the persons therein employed have ready and available means of ingress and egress by not less than two shafts, slopes or outlets, one or more of which may belong to another coal mine or colliery:

"*Provided*, That a second opening can be had through coal; but if a tunnel or shaft will be required for the additional opening, work upon the same shall commence immediately after the passage of this act, and continue until its final completion with not less than three shifts in each twenty-four hours, and as many hands to be employed as can be put to work to advantage, the inspector to be the judge as to the least number of hands engaged per shift; this section shall not apply to opening a new coal mine or colliery, nor to any working for the purpose of making a communication between two or more shafts, slopes or outlets, so long as not more than twenty persons are employed at any one time in said new mine or working;" . . . . .

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* *Owner* is defined to mean the "immediate proprietor, lessee or occupier;" *agent*, to mean a "person having, on behalf of the owner, the care or direction of a coal mine or colliery, or any part thereof."

† *Working* is probably intended to mean *mining* coal.

Special provision is made in Section 4 for the making of openings on or through lands owned by other parties. (See last chapter of this report, Mine Laws, &c.)

It therefore appears that not more than twenty persons can be employed in *opening* a new mine until a second outlet is made, and that no mining (working) shall be prosecuted in any colliery until a second outlet is obtained, which outlet shall be separated by one hundred and fifty feet of "natural strata" from the first opening.

If in opening a colliery by slope, the airway be sunk one hundred and fifty feet from the slope (both being ventilated by brattice or air-pipes to avoid the cost of driving cross-headings of such length) or if the airway is driven up from the bottom, the mine is then apparently provided, in accordance with the law, with two outlets, and any number of miners and laborers may be employed in the workings,—but the act was evidently not intended to be so construed. Two outlets located only this distance apart with workings opened out on both sides of the main slope would often be no better than a single outlet.

When the opening is made by shaft, the interests of the operator are usually best served by locating the second opening at a considerable distance from the first.

The law specifies that there must be "in communication with every seam or stratum of coal worked . . . . at least two shafts or slopes or outlets . . . . "This may be variously construed: thus, if in a colliery opened by slope or shaft with two outlets, the workings are extended to a second coal bed by tunnel, these tunnel workings are certainly "in communication" (through the tunnel) with "two outlets," but there is only one communicating passage and the tunnel workings are completely isolated in case of accident closing this tunnel,—but the intention of the Act is certainly to provide two outlets from such workings, which it apparently does not, if literally construed.

Again, suppose an inside slope is sunk and workings opened out which are more extensive than the old workings; these workings may be connected with the other portions of the mine by no other outlet, but if the mine has two shafts or slopes or outlets to the surface the requirements of the law are apparently fulfilled. An accident closing this inside slope (and its airway, which is usually only a few yards distant) entirely isolates those employed in the lower workings. Whether these contingencies were considered in framing the law; whether it contemplates treating *each such sub-division* of a colliery as *a separate* "coal mine or colliery;" whether the plain import of the act, to provide two outlets for the men *in every case*, is to be accepted instead of a literal construction of the wording of this clause,—these are legal problems which can only be solved by judicial decisions.

They have already been decided in part by the position taken by some of the inspectors, that not more than twenty men shall be employed in opening any new extension until a second outlet is obtained from it into other parts of the old workings, and this construction of the import of the act has been partly sustained,—but to place the matter beyond the pale of dispute and uncertainty, it is evident that the act should be so amended that all its provisions may be clearly expressed, and as easily understood by the practical miner as by the judge on the bench.

A decision defining the construction to be placed upon this section of the Act was rendered by Judge Rice of the court of common pleas of Luzerne county, April 4th, 1881, in the case of *Commonwealth ex rel. Williams, mine inspector* vs. *Haddock et al.*, which was an application by the mine inspector for an injunction to restrain defendants from employing and permitting persons to be in the first and third seams of coal, at the Dodson colliery at Plymouth, for the purpose of working* therein *at the same time* any person or persons were at work in the fifth seam. This fifth seam was connected with the other workings (which were provided with two outlets) by only one passage.

The injunction was granted. The opinion of the court can be found in the Report of the Inspectors of Mines for 1881, pages 151 to 159.

The second openings are frequently used as upcasts,—a fan being placed at the top,—and the main winding shaft forms the downcast or intake. When the breaker is built over the shaft, the danger to those employed below in case the breaker catches fire, is terrible. In some mines when the miners and laborers are scattered through workings several thousand feet or even one mile distant from the outlet, the smoke from the burning breaker would certainly reach and overpower a large number before they could find their way to the outlet.

The fan might be promptly stopped to keep the smoke from entering the shaft, but this would be a dangerous expedient at fiery mines, as the air would reach the explosive stage at many points in a few minutes, and before the miners escaped up the ladders of the air shaft, or before they even reached it, explosions might occur that would destroy both life and property.

This difficulty might be partly overcome by the adoption of fans capable of being quickly reversed; but fans of this class are not efficient, because when the current is reversed the gangway doors are blown open, the air travels through by the shortest cut, and some splits may receive such a small quantity of air that the gas quickly becomes explosive. In the excitement attending an alarm of this kind, lamps will almost surely be left by some miners or laborers in working places, where they will fire any gas that may accumulate or be blown down upon them from adjacent workings.

Another circumstance which must always greatly decrease the miner's chance of escape, is the fact that the way to the second opening often leads through comparatively unused, and to many almost unknown, passages; and that even those knowing the way tolerably well may not know it well enough to find it readily when laboring under great excitement, or when deprived of the guidance (if the fan is stopped) the ventilating current always affords.

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* Mining?.

At shaft collieries the miners are commonly lowered and raised on the cage, ten or less at once, but at some collieries they have access through airways or breasts worked up and holed through to the surface.

At slope collieries on steep dips the men are raised and lowered in a mine car, double chains being attached in compliance with the law.

The law provides that "in no case shall more than ten men ride on any wagon or cage at one time in any of said mines," and this passage has been construed by some to permit the raising or lowering of twenty men when two cars are run in one trip on the slope—ten in each car—but the intention of the Act is evaded by such a construction, and the mine inspectors have very properly insisted upon a different interpretation, viz: That not more than ten persons shall be raised or lowered at once.

When the dip is less than twenty-five degrees the miners commonly walk down the slope or some traveling-way nearest the breasts in which they are working.

Traveling-ways in slopes are located on one side of the hoisting compartments; sometimes the pump-way is also used as a traveling-way; sometimes the traveling-way is on the opposite side, and is separated from the hoisting compartments by a vertical prop set in each frame of timber.

When the dip is slight the bottom sills, when filled in between with mine *débris*, break the descent into convenient steps; but when the dip is steep, and the timbers not set close together, a plank stairway is built (See Fig. 14,).

Air shafts used as second outlets when not provided with winding machinery, or when such machinery is not kept in readiness for immediate use, are fitted with ladders. These are cheaper and offer less resistance to the ventilating current than a stairway, but there are many obvious reasons why the latter should be preferred for second outlets.

The man-engine is not used in the anthracite regions, but there can be no doubt but that at some of the deep collieries of the future they can advantageously replace the method of raising and lowering the men on the cages used for raising coal.

One of the most powerful man-engines ever constructed is shown on Atlas sheet XXIII, designed by Mr. E. D. Leavitt, Jr., for the Calumet and Hecla copper mines in Michigan. It is geared to make five double strokes per minute and therefore lands five men at the bottom every minute, or three hundred in one hour,—to this must be added the time occupied in the descent of the first man, which necessarily depends upon the depth.

The chief advantage of the man-engine for use in the anthracite coal fields would not be the time saved the miner (as frequently stated) but will I think be found to be the facility it offers for the ascent and descent of persons independently of the winding machinery.

Coal could be raised and the breaker started without waiting until the miners have descended, and the winding of coal need not be suspended during or near the close of the day to allow the miners to ascend.

This advantage is frequently secured by the addition of a cage in a separate compartment used exclusively for raising and lowering men, tools, &c.

At large collieries the time consumed in raising and lowering the miners and other employés may exceed two hours, and as our mines become deeper and the average number of men employed inside is increased to five or six hundred the loss of time from this cause will become more and more serious.

The man-engine, however, offers no facilities for the transportation of miners' tools and supplies,—this is an almost fatal defect,—but on the other hand it is claimed that it may be also used to operate pumps or for the transmission of power. While this is true, it is also true that a proper regard for the safety of those who use it as a means of ingress and egress, should prohibit such a combination.

An independent cage set apart for the raising or lowering of men, tools, and mine supplies will probably give the best results at anthracite collieries. When the output is to be so large that two winding compartments will be insufficient, provision is now made for two additional compartments, (and a large air-way is at the same time secured) by sinking shafts of the large cross-sectional area mentioned in chapter IV.

When this is done the time necessarily occupied in raising and lowering the employés will have little influence on the output, even at collieries raising twelve or fifteen hundred or even two thousand tons per day.

Collieries putting out one thousand tons of coal per day (the Wilkes-Barre district can boast of several such) require not only rapid, but uninterrupted, winding for at least eight hours.

A car load of coal as it comes from the mine will yield about two tons of merchantable fuel; therefore, five hundred cars of coal must be raised to make an output of one thousand tons. To do this in eight hours (nett) the cages must *average* 62½ runs per hour, and this result cannot be accomplished unless the maximum capacity of the winding plant is at least 70 runs per hour, or one winding including car shifting, signaling, etc., in about *fifty-one seconds.* The actual time of winding and seating the cage on the wings would be limited to about *thirty-five seconds.* If the depth is 600 feet the speed in the middle of the run must be about *forty feet per second.*

As the *speed of winding* cannot be increased with safety much beyond forty feet, it is evident that at deeper shafts more time must be given to each winding, and eight hours will not be sufficient, so that if coal is to be raised for nine or ten hours, little time is available for raising and lowering the men.

At the Pottsville shaft one winding is easily made in forty-five seconds, the depth being about sixteen hundred feet, an *average* speed of *thirty-five feet per second*, which shows a speed of at least *forty to forty-five feet per second* in the middle of the run.

English engineers do not seem to have attained any better results in practical winding than can be shown at this Pottsville shaft, and at some of the Wilkes-Barre mines, where the machinery is lighter and the cost of the winding plants much less, and it is extremely doubtful whether any mining district possesses winding plants as cheap and as efficient as some of our anthracite collieries.

Chapter XVIII.

Drainage and Pumping machinery.

Water that can be caught by a water level gangway opening by drift or tunnel to the surface, is conveyed directly from the mine, but on steep dips it is frequently impossible to hold the water at this level. The chain pillar often falls or fissures open in it which let the water down into the lower workings.

In the earlier days of anthracite mining, an effort was made to hold the water in each lift by leaving thick chain pillars on the lower side of the gangway, but on steep dips and in thick seams these pillars were necessarily very large, and involved the loss of such a large quantity of coal. This practice has now generally been abandoned and the water (except that caught on the water-level gangway) is allowed to find its way into the lower workings.

By the adoption of this plan, the necessity for keeping open the gangways on worked-out lifts disappears, and the chain pillars may be worked through as soon as the gangway above is no longer needed for mining or ventilation.

In some districts the coal is so soft and broken throughout by clefts and fissures, that it is absolutely impossible to keep the water up.

As the water-level gangway generally catches more water than any lower gangway, especially in very wet weather when a large amount of surface water leaches through into the upper workings, it is considered especially desirable to confine this water to this level so that it may be conveyed from the mine without pumping, and to insure this, very thick chain pillars are often left below this gangway. But these measures often prove futile and nearly all the water ultimately finds its way into the lower workings.

In some portions of the anthracite regions, especially in the Hazleton group of basins, the surface above the workings sometimes consists of swampy land, very imperfectly drained by sluggish streams. In wet weather these swamps are partly converted into ponds, and overflow areas containing the outcrops of the seams worked,—the water then flows in large quantities into the workings.

Large expenditures have been made to drain such areas by ditching, and in some places to divert the course of a stream by a ditch, carrying the water around the area underlain by the outcrop or opened by crop-falls.

In some parts of the Wyoming district, where the coal has been worked until the subterranean outcrop beneath the water-logged gravels now filling the ancient water channel, stopped further operations and at other places not yet worked, the danger of sudden flooding should be most carefully guarded against. This old river bed may, in places, be one hundred and fifty or two hundred feet deep, and the water under this head, (for all gravel beds are filled with water,) equal to a pressure of sixty or seventy pounds to the square inch, might flood a mine so quickly that none could escape.

The danger from swamps on the surface has been effectually met by the drainage established, but the risk of serious and sudden flooding from this source has never worn the serious aspect presented by some of the prairie mines of the western States.

An accident similar to the terrible catastrophe of February 16th, by which about eighty persons were drowned by the flooding of the Diamond mines at Braidwood, Ill., would not be likely to occur in the anthracite regions. This disaster was occasioned by the water covering the prairie finding its way through the surface at a point where the workings had been pushed to such an extent (long-wall system) that a break occurred, allowing the overlying strata to settle down. The workings were only about ninety feet below the surface.

The law provides against the risk of flooding from water standing in old workings, by requiring that "bore holes shall be kept twenty feet in advance of the face of each and every place, and if necessary, on both sides, when the same is driven towards or approaching an abandoned mine, or part of a mine suspected to contain inflammable gases, or which is inundated with water." (See Section 9.)

Casualties from flooding by surface water are usually caused by a crush or crop-fall opening communication between the bed of a stream and the workings beneath. Thus the accident at Pine Brook and Fairlawn workings Sept. 4th, 1878, (see Report of Mine Inspectors 1878, page 158 to 163) and at Harleigh colliery, June 20th, 1877, (Inspector's Report for 1877, page 181, &c.,) both had this origin.

The cave-in at Harleigh, extended to the pump-way, and it became necessary to remove the pumps, and as the water from Black creek was flowing into the mine, it soon filled up. The water from the creek was finally diverted from its channel and conveyed in a flume over the disturbed area.

The pillar between this colliery and the Ebervale colliery workings was not thick enough to stand a pressure of more than three hundred feet head of water, and it became necessary to draw off the water from Harleigh to prevent the possibility of a most serious disaster. Prior to the crush and in anticipation of the exhaustion of Harleigh a heavy timber dam had been built to hold the water back, but when the Harleigh pumps were abandoned and it was known that the water would soon reach a height of over three hundred feet, it became apparent that the dam was useless because the pillar being only about forty feet thick was not considered strong enough to stand this pressure.

The water was drawn off through a series of six three-inch holes. Each hole was driven four or five feet into the pillar and a pipe inserted, which was held in place by a heavy prop, 12′ ×12′, as shown by Fig. 50.

The hole was then drilled through the pillar. A mortice was cut in the side of each prop so that the water could be shut off with a key.

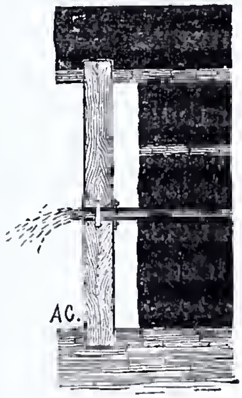


Fig. 50.

These holes discharged over 1800 gallons per minute, to remove which pumps of 2206 gallons' capacity were added to the pumping plant. A full description of the pumps and other details may be found in the report of the Inspector of Mines for 1877, pages 193 to 195 and 201 to 206.

Page plate No. 37 shows a map of the boundary pillar between the two workings and a vertical section across the basin, reproduced from the Mine Inspector's report.

I have introduced this brief resumè merely to emphasize the necessity for strong barrier pillars between adjacent workings, not only to prevent the risk of sudden and disastrous flooding, but also to obviate the necessity of draining an abandoned mine that another may continue working.

In attempting to drown the Kehley Run fire by flooding the same difficulty was encountered, and there are many localities at present in exactly the same condition.

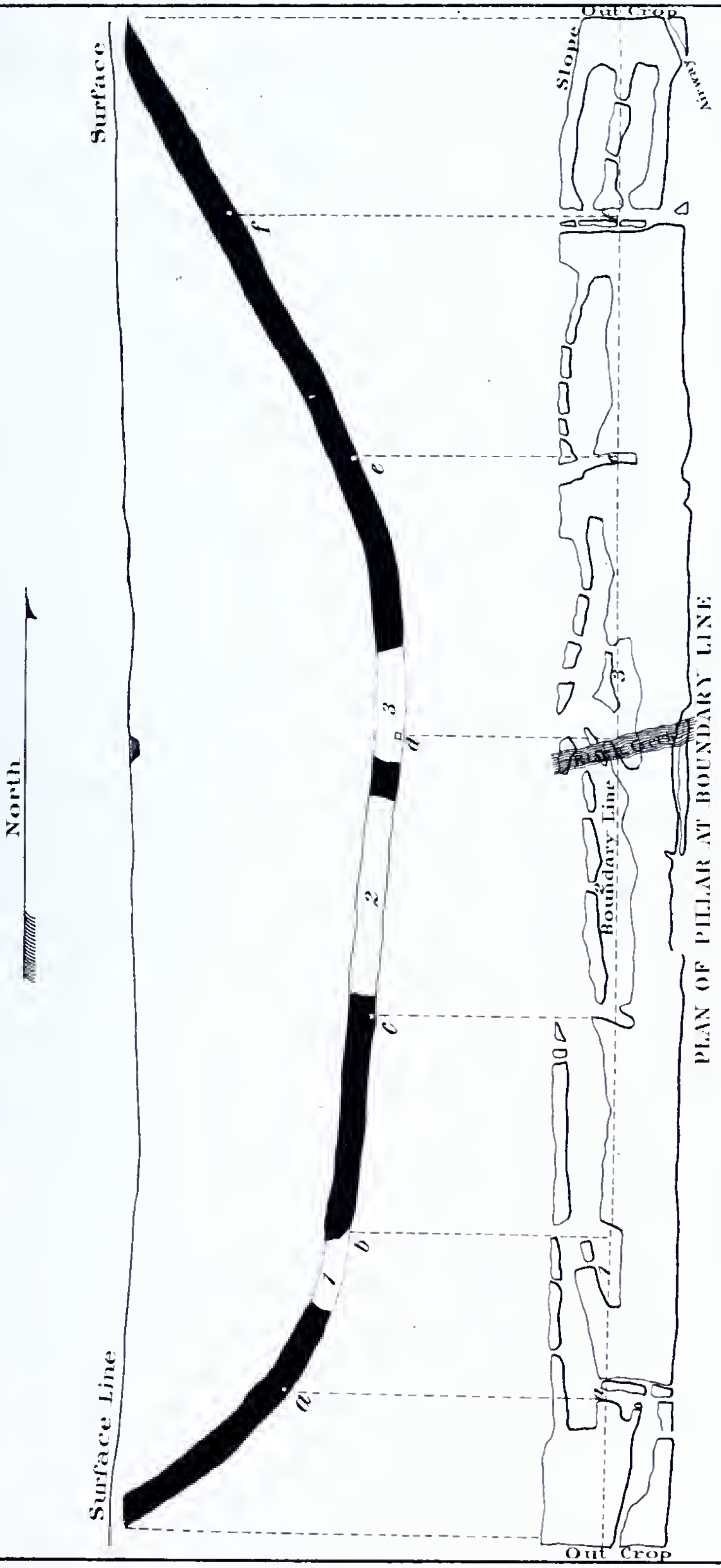
When the property line of a mining company runs nearly parallel to the outcrop there is always a strong temptation to sink down to the line, and to open and drive the gangways as close to the line as possible, thereby greatly increasing the output. When this is done, (and it has been done at many places,) the coal lying to the dip of these old workings is in constant danger of flooding, and workings on the adjoining tract cannot safely be extended in that direction unless the old workings are unwatered.

Bore holes twenty feet long do not furnish sufficient safeguard in a thick seam or on steep dips.

Within the last few years several of the larger operating companies and many of the smaller operators, have begun to leave thick barrier pillars at the property lines, and when these are pierced for ventilation, or to furnish the second outlet required by law, the passages driven through them are made as small as possible for three reasons: 1st. So that the pillar shall not be appreciably weakened, 2d. So that a strong reliable dam can be quickly and cheaply built to isolate one working from the other, and 3d. To prevent an explosion in one working from doing serious damage in the other.

These barrier or division pillars are from one hundred feet to as much as one hundred yards thick. one half this thickness being left on each side of the property line. They occasion no real loss of coal as they can be mined out when

From Report Inspector of mines 1877.



SECTION AND PLAN OF BOUNDARY BETWEEN EBERVALE AND HARLEIGH WORKINGS.

the mines on both sides are to be abandoned. If one mine is exhausted before that adjoining it, the operators continuing work can in some cases insure the integrity of the pillar by purchase,—but a safer plan is most assuredly for each party to leave sufficient thickness to insure absolute safety without reference to operations on adjoining tracts.

Boundary pillars are not only valuable in separating adjacent workings so that danger from water, fire, and explosions is greatly reduced, but they also constitute a most important barrier to the extension of a squeeze or crush, so that one mine can be robbed out and abandoned without involving the adjacent mines in the crush which inevitably follows this procedure.

In lift mining the sump is generally excavated in the coal beneath the gangway level. It is certainly advisable to have a sump of sufficient capacity to hold at least as much water as will accumulate in forty-eight hours or more, so that there shall be ample time for repairs, changes, or alterations in the pumping machinery, but as this is not always possible, and even when possible is not always considered essential, we find some mines provided with totally inadequate sumps,—sumps that will overflow in a few hours (at certain seasons of the year).

In flat workings there is often great difficulty in providing sufficient sumpage.

The drains, ditches, or gutters have already been described in the chapter on gangway driving. They need thorough cleaning at regular intervals to insure free passage of the water.

Mines in the Wilkes-Barre region are commonly known as "dry mines," that is, the water is rarely troublesome and the deep workings are often very dry.

In the Lehigh district and in some parts of the Schuylkill region the mines are very wet. In some parts of the Lehigh district *an average* of about *nine tons of water* is pumped for every ton of coal produced, and at some collieries more than twenty tons of water have been raised for every ton of coal.

At some of the deeper shaft collieries of the Schuylkill region and in the Wyoming deep mines the amount of water to be pumped is by comparison insignificant, often falling below the tonnage of coal raised. However, a colliery producing eight hundred or one thousand tons of water per day, (as many of the mines of the Wyoming district,) which has to be raised a vertical height of six or eight hundred feet, requires pumps of no mean capacity.

It is not within the province of this report to go into a general discussion or description of pumping machinery, but there are some features pertaining to the unwatering of anthracite mines, which being peculiar to anthracite mining, may properly be discussed here.

Two classes of pumps are in common use; the pole or rod plunger pumps of Cornish "pit work" pattern, and inside steam pumps.

The arrangement of the rods, the method of strapping them, of attaching the plunger rods, of providing "checks or catches," are similar in every respect to the practice in other deep coal mining districts.

But the method of applying the power is generally different. At most mines these pumps are operated by a fly-wheel single high-pressure engine, geared down to the required number of revolutions (five or six) per minute; the driving pinion or shaft carries a crank which is connected with the pump-bob by a connecting rod.

This method has generally been adopted because any old engine could be very cheaply obtained (or one might be on hand) and geared to furnish the required power.

Within the last few years the so-called "Bull" engine has been introduced at several collieries.

Atlas sheet No. XII, shows in detail a Bull pump at the Exeter shaft; the arrangement of the pit work is shown on a smaller scale by Atlas sheet No. II. This pump is fifty-five inches in diameter by ten feet stroke.

The drawing shows the details so plainly that a lengthy description is unnecessary.

The average working speed is four to five strokes per minute, but while it is claimed by the engineer in charge, that the pump can be run up to eight strokes, I do not think it can safely be made to exceed six and a half strokes on a lift of this height.

The pump compartment is arranged to accommodate two such pumps (See Atlas plate No. II), both of which to discharge into a single column pipe, but up to the present time one pump has been more than sufficient. This pump was built at the G. W. Snyder works at Pottsville.

At the Pottsville deep shaft the water is nearly all caught in workings on an upper seam at a depth of about three hundred feet, and removed through a pump-slope by a Bull pump of construction similar in every way to that at Exeter, except that the bed-plate is triangular, inclining the steam cylinder to the angle of inclination of the pump-slope. It is doubtful whether such large cylinders will wear well when inclined at a considerable angle, and some engineers believe they should always be provided with back-piston rods; but the experience of many of our most prominent engineers has been such that they do not consider the addition of a back-piston rod necessary or even advisable.

That the Bull engine gives a much higher duty than other methods of driving-rod pumps now in use in the anthracite regions is not admitted by many engineers.

The following reasons probably summarize the facts and conditions which have prevented the adoption of this form of engine at many collieries:

1st. Their cost.

2d. Engines already owned by operators that may be utilized for pumping.

3d. The space they occupy.

4th. Fear of serious damage to steam cylinders, etc., from breakage.

5th. Time necessary for repairs or renewal of steam cylinders or valve chambers, (in case of accident.)

6th. Risk to steam cylinder, etc., from fire in shaft.

Cornish engines with the steam cylinder at one side of the shaft, the power being transmitted through a walking beam, have not been adopted in the anthracite regions.

Steam pumps of almost all forms, sizes, and makes, are to be found inside the mines throughout the anthracite regions. The number of collieries at which no steam pumps are used inside is comparatively small.

To show the great variety of direct-acting steam pumps in use I subjoin a list which represents only a small number of the different makes:

The Cameron, Blake, Knowles, Allison, Clark's Duplex, Salkeld's Duplex, Thatcher, Bradley, Carter's, Niagara, Guild and Garrison, Albright and Strohl, &c., and in addition to these, nearly all of the large operating companies have special forms built after designs by their own engineers, often at their own shops, and nearly every machine shop in the region manufactures some special style.

They are nearly all plunger pumps; piston pumps having been discarded as unsuited to mine work, and especially unsuitable when the water is as strongly acid as at most anthracite collieries.

The principal objection to the use of steam pumps under ground is the exhaust steam. At some collieries the exhaust is conveyed to the surface through a pipe laid for that purpose, at others it is conveyed into the sump, and in some cases it is turned into the upcast airway.

When the pumps exhaust into the sump it is often found that the whole body of water is heated to a comparatively high temperature, raising the temperature of the mine and increasing the humidity of the air to such an extent that the mine timber decays with ruinous rapidity, and at some collieries the roof and coal on the airways, gangways, and travelingways is softened and becomes both troublesome and dangerous.

When the exhaust is conveyed to the surface through a pipe of considerable length, trouble is caused by condensation of steam in the pipe and also by the radiated heat.

The practice of conveying the exhaust into the upcast is often ruinous to the walls, roof, and timbering of that passage.

These difficulties in the use of steam pumps under-ground are supplemented by the following additional objections:

1st. Loss of steam by condensation.

2nd. Uselessness of pump when drowned out, either by breakage or sudden flooding.

But as pumps are often needed in locations where rod-pumps cannot be used, steam pumps are absolutely necessary at many collieries.

The loss from condensation can be reduced to a small amount by properly covering the steam pipes; the risk of being drowned out can be removed by having a surplus pumping capacity and reserve pumps in readiness in case of breakage; the principal difficulty is therefore with the exhaust.

If this cannot be conveyed to the surface by a short route, there seems but one alternative,—either to endure the damage occasioned by heat of the exhaust or to substitute compressed air for steam as the motive power.

This can doubtless be done with advantage at many collieries, but in some cases the cold produced by the exhaust will be almost as troublesome as the heat from steam.

A few collieries are not provided with pumping machinery, but the water is raised by the winding machinery. This method is adopted where the amount of water is small and the output of coal is not large. At slope collieries a well connecting with the sump, or the sump itself, is located beneath the landing at the lowest level, and the slope tracks are continued down below the level of the water.

The water is raised in a tank, usually of boiler iron (often an old boiler shell), provided with large flap valves and mounted on a carriage.

When the slope is not raising coal—early in the morning, at noon, late in the afternoon, or perhaps at night—this tank is run upon the slope and the water raised. The valves are so arranged that the tank fills when lowered into the water, and on reaching the surface may be quickly emptied by means of a lever, opening the valve.

At shaft collieries the water may be raised in a tank attached beneath the cage, or in tank cars of about the same size as a coal car, which after being filled from the sump by a small pump, are run on the cage and raised like an ordinary mine car. This is the plan now used at the Pottsville deep shaft.

The plan sometimes used in France of raising water (at shaft collieries) in a tank in one compartment and coal in the other has not been adopted in the anthracite regions.

The principal difficulty experienced with all forms of pumping machinery throughout the anthracite regions arises from the corrosive action of the mine water. At many collieries the water is so highly acid that it will eat completely through a twenty or twenty-four pound rail in a few weeks.

To prevent rapid destruction all parts exposed to the action of the water are made very thick, the working parts are often lined with, or entirely made of variously proportioned (brass, gun-metal, etc.) composition metals, and the column pipe, elbows, branches, etc., are often lined with wooden staves, carefully coopered like barrel work.

Wooden column pipe is in use at a number of collieries, and doubtless has many advantages over the cast-iron pipe commonly used. It may often be substituted for the latter when pipe of a moderate size is required, but for large pipes cast-iron is always preferred.

Wooden pipes are made by boring out white pine logs, then turning the outside, leaving the pipe two or three inches thick. After being tested by hydraulic pressure they are tightly banded with an iron band, which is wound spirally around them by steam power. Additional coils are wound around the ends to brace the joints. After being coated with coal tar and asphaltum to protect both the wood and iron, they are tested by hydraulic pressure, and if found perfect are ready for use.

These pipes are made in segments and are united by a socket joint.

The great bulk and weight of heavy steam pumps used underground necessitate heavy bed-plates, and these as well as other massive parts are commonly made in segments to be bolted together, so that they can be more readily lowered into the mine and transported to their location underground.

Timber seats are common enough, but stone foundation pillars, and sometimes combined stone and timber seats are preferred.

To give satisfactory results steam pumps for use in anthracite mines, must have:

1st. Ability to resist corrosive action of mine water.

2d. Great strength of body and all working parts.

3d. All working parts, valves, etc., readily accessible and capable of being quickly renewed or repaired.

4th. Simplicity.

5th. Large water-ways, and valves.

6th. Must be readily adjusted to run at any number of strokes per minute (below the maximum).

7th. Regularity of action.

The Allison form of steam valve gear in which the movement of the valve is regulated by a cataract, is considered by a large number of engineers better than any other style for inside pumps.

While at many collieries two or more very large rod pumps, as well as several large steam pumps are necessary, the introduction of large pumps is generally restricted to collieries at which two or three pumps of modern size would not be sufficient ; in other words, it is generally considered better policy to unwater a colliery with two or three pumps of moderate capacity than to depend on one large pump.

The latter may do the work with less steam, but in case of breakage there is no means of keeping the water down until repairs can be made, while if two or three pumps are used, the mine can generally be kept from flooding while one is undergoing repairs, by working the others at increased speed.

The *hydraulic system of pumping* now used in some of the deep mines on the Comstock lode in Nevada, has not been adopted in the anthracite region. It does not seem improbable that this system may be advantageously adopted at mines in which the exhaust steam is very troublesome. The cost of the plant, the loss of power, and the cost of pumping are undoubtedly greater by this than by any direct method, but the advantage which it secures in abolishing the exhaust may more than repay the increased expenditure.

Table showing the Performance of Pumps in the Lehigh District in 1876.

LOCATION OF COLLIERIES.	Slope number.	Number pumps and pumping engines	Horse-power of pumping engines.	Pressure of steam per square inch	Diameter of pump in inches.	Length of stroke in inches.	Absolute capacity of pumps during the 24 hours.	Approximate number of gallons of water pumped per 24 hours.	Approximate number of tons of water pumped during the time worked in 1876.	Number of days worked in 1876.	Vertical height of column in feet	Number of tons of coal produced during the time worked in 1876.	Number of tons of water pumped to each ton of coal produced	NAMES OF PUMPS OR MAKERS.
Upper Lehigh, . .	1 2	3	234	75	12, 14, 16	48, 72, 72	4, 199, 040	834, 240	724, 863	233	181 240	167, 431	4. 32	Roberts' steam.
Do.	3 4	3	78	75	14, 4, 8	72, 10. 36	2 410, 560	282, 211	223, 024	212	86	92 517	2 41	Roberts' steam.
Eckley Coal R'ge,	2	2	100	75	12, 12½	72, 72	2, 016 000	1, 010, 720	640, 900	170	330	86, 337	7. 42	Thatcher & Bradly steam.
Do.	4	1	40	60	12	72	912, 960	888. 960	417, 778	126	210	38 298	10 91	Bradly steam.
Do.	5	1	70	75	12	48, duplex	1, 296 000	126, 600	17, 464	37		7, 404	2. 36	Roberts' steam.
Beaver's Brook, .	2	2	90	60	8, 14	72	2, 160, 000	545, 000	447, 260	220	130		5. 73	Allison & Roberts' steam.
Do.	4	2	80	60	8, 10	72, 48	1, 620, 000	330, 000	247, 431	201	370	121 200		Roberts' steam.
Jeanesville, . . .	4	1	50	75	14	72	3 456 000	720, 000	491, 446	183	283	104. 075	4. 72	Allison steam.
Do.	1	5	360	75	20, 12	3-120. 2-72	6. 882 000	2, 400, 000	1, 656, 120	185	230	77, 708	21. 31	4 plunger and 1 Allison steam.
Do.	5	2	100	75	14. 14	72, 72	3 456 000	1, 440, 000	988 264	184	229	106. 254	9. 3	Allison steam.
Lattimer, ...	2	2	120	80	17 17	48, double act'g	‡1, 929, 600	906, 336	425, 880	126	297	82, 000	5 19	Thatcher steam.
Stockton, ...	4 5	3	292	65	12, 18, 14	72, 72, 72	7, 885, 440	2, 419, 200	1, 967 014	218	261	104, 541	18 81	Thatcher & Allison's steam.
Yorktown, ...	5	2	150	80	14, 12½	72, 72	2, 304 000	1 073, 000	672, 336	168	276	127, 797	8. 87	Thatcher & Allison's steam.
Do.	6	1	100	70	17	48, double act'g	1, 086 048	763, 200	461 165	162	255	. .		Thatcher & Allison's steam.
Highland, ...	1	3	120	60	12, 10 8	72, 72 72	1 704, 960	1 704, 960	1 297, 236	204	319	99, 728	13.	2 Thatcher and 1 Cameron's steam.
Jeddo Oakdale, .	1	1	40	80	14	60	1. 152. 000	576. 000	292. 128	136	313	68 851	4. 24	Roberts' steam.
Do.	2	4	250	80	6, 6, 8. 8	36, 36, 36 36	3 312 000	1 656, 000	827, 718	134	205	60, 731	13 63	Albright & Stroh.
Tresckow, ...	6	1	90	70	16	96	864, 000	792, 760	589, 182	199¼	224	122, 977	4 79	Pole pump.

Nesquehoning, .	1 2 3	5	725	75	20, 16, 8, 12, 8	120, 72, 24, 72, 24	2,878,560	1,438,780	1,053,077	196¼	288	116,010	9.07	Carter, Allen & Co., L. & W. B. Coal Company, 2 Knowls' and I. H. Salkill.
Drifton 'C's C'K.'	1	2	350	70	12, 12	72, 72	2,217,600	1,104,000	729,915	177¼	237	136,647	5.34	Meyrick & Wren steam.
Woodside, . .	1	2	350	70	10, 10	96, 96	1,310,400	648,000	297,291	123	136	34,455	8.63	D. Clark & Co., Hazleton, Pa., steam.
Beaver Meadow, .	1 2	3	230	80	12	72	1,842,000	1,322,880	592,200	120	75 235	73,428	8.06	Reese's vacuum & Wren steam.
Summit Hill,	4	4	1,280	90	20, 20, 20, 20	120, 120, 72.72	†5,342,400	2,671,200	1,412,253	141¾	†620	66 264	21.31	Roberts' & Salkill, Carter & Allen.
Do.	5	1	50	90	16	72	907,200	576,000	384,492	179	265	84,521	4.55	Roberts' & Salkill steam.
Coleraine, . .	2	2	210	60	16, 10	72, 72	1,058,400	751,680	555,192	198	230	67,565	8.21	Wm. T. Carter & Co. steam.
Ebervale, ...	1 2 3	3	380	85	14, 12, 8	72, 48, 48, duplex	3,007,204	2,585,204	3,249,364	337	284	228,248	14.23	D. Clark and 1 Camron steam.
Harleigh, . .	1 3	4	315	75	12, 12, 13, 11½	60, 72, 60, 48	3,456,000	2,101,853	2,216,760	282¾	312	130,462	16.99	Allison steam.
Hazleton collier's,	1	5		110	13, 11, 10, 14	60, 60, 60, 60	4,497,120	869,280	*A353,702	109 1-10	.	D148,386	12.08	1 piston pump & 4 pole.
Laurel Hill coll's,	5	7	.	80	18, 16, 15, 12½, 10	84, 84, 72, 72, 96	5,493,600	2,533,080	B1,103,054	116¾				2 duplex, 2 Thatcher, and 3 pole.
Sugar Loaf coll's,	1 2 3	6		65	12½, 13, 12, 12½, 16	72, 84, 72.72, 84	3,391,200	1,458,360	1,414,400	260	. .	63,859	22.15	2 Thatcher, 3 pole, and 1 piston.
Hazleton collier's,	3	8		65	13, 11, 10, 14	60, 60, 60, 60	1,834.560	1 084,020	C335,569	83				4 duplex st'm & 4 pole.
E. Crystal R'e col.,	4	2		70	12½, 10	72 72	576,000	532 800	189,758	95½				2 Thatcher steam.
Crystal Ridge col.,	2	1	.	45	18	72	907,200	56,880	23,108	109	. .	82,005	7.99	1 pole pump.
Cranberry coll's,	1	4		65	14, 14, 14.15	84, 84, 84, 72	2,725,920	1,088,640	442,649	109				3 pole and 1 Thatcher steam.
Mt. Pleasant, . .	2	3	.	70	12, 14	96, 84	2,108,160	564,480	242,075	115	. . .	38,344	6.31	2 duplex steam & 1 pole.
Hollywood, ...	2	2	. .	70	14, 12½, 18	60, 96, 84	2,808,000	643,980	309,858	129	. .	63,691	4.86	1 piston, 1 Thatcher, and 1 plunger.

*A, B, C are added together and divided by D. †Two lifts. ‡No. 1 pump; No. 2, 1,728,000.

The quantity of water lost through the valves has not been taken into consideration.

This is a subject that can only be decided by each engineer in reference to the special conditions and requirements at each colliery, after careful inquiry into the differences in first cost, fuel, repairs, attendance, the duty performed, and the money value of the advantages gained by the abolition of steam underground.

This applies equally well to the proposed plan of running pumps by power transmitted from the surface by means of wire ropes.

The accompanying table from the Report of the Inspectors of Mines for 1876, compiled by Mr. T. D. Jones, shows the performance of 25 rod pumps and 72 steam pumps in the Lehigh district. While it apparently contains some erroneous figures, it is nevertheless valuable in showing the large amount of water pumped in this district, and the large reserve pumping capacity provided.

Chapter XIX.

Ventilation and Ventilators.

It is beyond the province of this report to enter upon a discussion of the principles and theories upon which all ventilating systems are dependent. The current mining-engineering literature of the day is filled with articles treating not only of the principles but also of the details of mine ventilation, and several pamphlets and books—mostly English, but that are for sale by booksellers in this country—lately published, thoroughly and clearly define these laws ;

"Friction of Air in Mines," by J. J. Atkinson, published by Van Nostrand, "Coal, Mine Gases, and Ventilation," by J. W. Thomas, London, Mr. Hermon's "Prize Essays on the Prevention of Accidents in Mines," London, 1874, are standard reference works.

The correspondents' column of the Shenandoah Mining Herald furnishes a medium through which the miners and mine bosses of anthracite collieries are constantly interchanging their opinions and recording their experience,—a large number of the communications bear directly upon this subject, and contain information valuable to every miner. The chief objection to open columns in which all communications are published, is that many are unable to separate the chaff from the grain, and frequently accept the opinions of others as demonstrated facts.

In the Wyoming district, especially in the neighborhood of Wilkes-Barre, the best means of securing proper ventilation is the most important mining problem to be solved. Great trouble is also experienced in dealing with explosive mine gases in the deeper portions of the Schuylkill basin.

In the Wilkes-Barre district, workings on the lowest level are sometimes opened with two airways and a gangway, each driven twelve feet wide by six or eight feet high, to drain off the gas, and it is often necessary to let the gangway and airways stand for several months before workings can be opened out.

When two airways are driven, the gangway is located between them, with eight or ten yard pillars between, and the intake air is divided into two splits, one passing through the new workings and the other out through the airway next the solid.

Throughout this district the airways are usually from seventy to over one hundred feet of cross-sectional area.

In the Lehigh region they are frequently made just large enough to satisfy the requirements (20 and 25 sq. ft.) of the Mines Ventilation Act.

The plan usually pursued in opening new workings is to drive an airway and gangway together, with a pillar of six to twelve yards between. When the bed is thick and dips steeply, the airway is driven in the coal over the gangway, and the airway coal is then loaded in cars standing on the gangway. Cross-headings are holed through between the two passages at intervals of ten to twenty yards.

The working face of the gangway and the airway are ventilated either by means of a brattice or brattice cloth hung near the cross-heading to throw the air-current into the face, or by means of a small blower (fan) and wooden air pipes (See Figs. 55 and 56.)

Cross-headings.—These are generally driven as small as possible; three to five feet wide and four to six feet high are the common limits. As they are for temporary use only, and must be closed by doors, brattice, or masonry stopping, they are made only large enough to pass the required amount of air.

Doors.—Wooden doors, so hung (on a post, or in a frame sill) that they are self-closing, are in use throughout the anthracite region. The space around the door-sill is planked or boarded up, or closed with masonry to prevent leakage, and a piece of brattice cloth or some similar material is often tacked to the bottom of the door to lessen the leakage. They are hung to open against the air-current.

The number of doors necessary depends principally upon the method by which the breasts are ventilated.

All of the main doors are opened and closed by a door-tender (*"trapper boys"*) whose sole duty is to open the door for the passage of a trip of cars, and to see that it is promptly closed after the passage of the cars.

At some collieries the number of doors is very large, and several devices have been invented to either open and shut the door automatically, or to enable the driver of the trip to open and close it by means of ropes, levers, etc. None of these have been adopted in the anthracite region. The law requires a door-tender for every main door, and also provides that the doors shall be so hung that they will be *self*-closing, therefore these appliances could not be legally used,—but there are other objections that would effectually exclude them from nearly all anthracite collieries, viz: 1. The door may not open promptly and the mules and cars may suffer; 2. The door may be closed by the driver before the cars have all passed through; 3. A lump of coal on the roadway may keep the door from closing, or prevent it from being opened; 4. The automatic devices are complicated, expensive, and apt to get out of order or to get clogged up with coal and dirt.

Stoppings.—Masonry is generally used for stoppings in fiery mines, but temporary stoppings, and stoppings in non-gaseous mines, are often built of boards or plank. In flat workings in thick seams massive masonry walls are often necessary to close up old workings. These walls are built of slate and rock picked out of the gob, and are not infrequently from four to six feet thick at the base.

Stoppings closing airways of ordinary size are made from one to three feet thick.

Many engineers believe that all stoppings should be made very heavy in order to resist the shock from an explosion of gas and confine the effects of the explosion to the district or split in which the accident occurs: others hold that this practice only confines the gas so that the effect of the explosion is concentrated on the passageways (airways, etc.) remaining open, and that the damage done and the danger to life are greatly decreased when the stoppings blow out, as these new openings act as vents or safty valves, and may allow the force of an explosion, which if confined would do great damage, to distribute itself almost harmlessly throughout the mine.

Brattices.—The customary method of building brattices is shown by Page Plate No. 9. A series of posts are placed along the gangway or passage, and a board partition built, care being taken to cut the top boards to fit inequalities in the roof; waste is thrown against the bottom board to close any openings caused by irregularities in the floor.

It is impossible to make an ordinary brattice even approximately air-tight. Long brattices always leak a large quantity of air. Better results can be obtained by using grooved-and-tongued lumber, than with ordinary boards.

Fig. 51 shows the method of ventilating new workings opened by tunnel on the Wharton bed at Hazleton. The air is carried in through an airway bratticed off on one side of

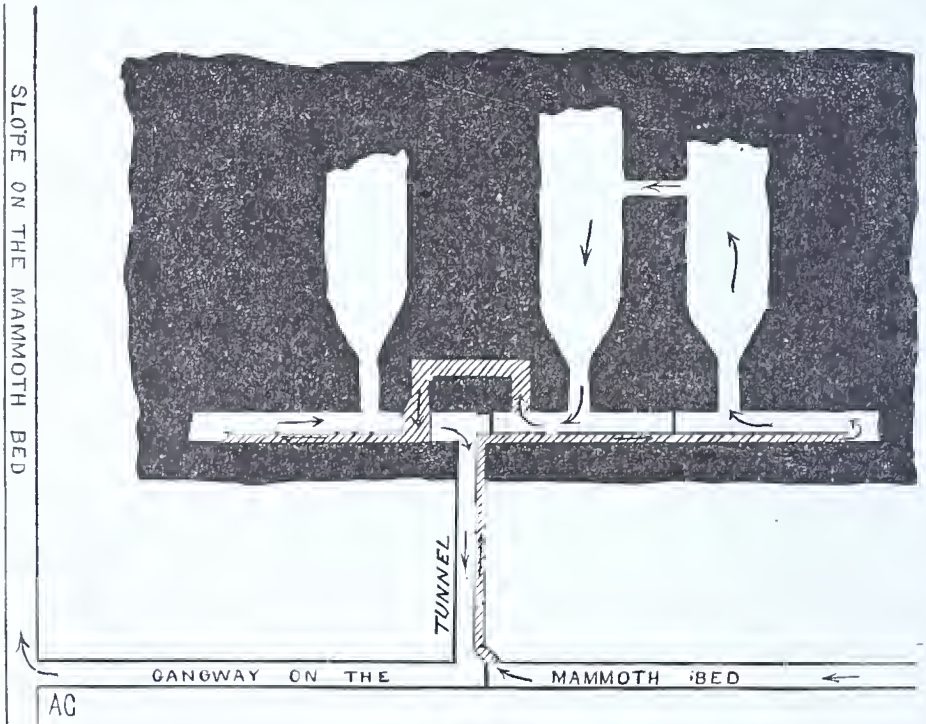
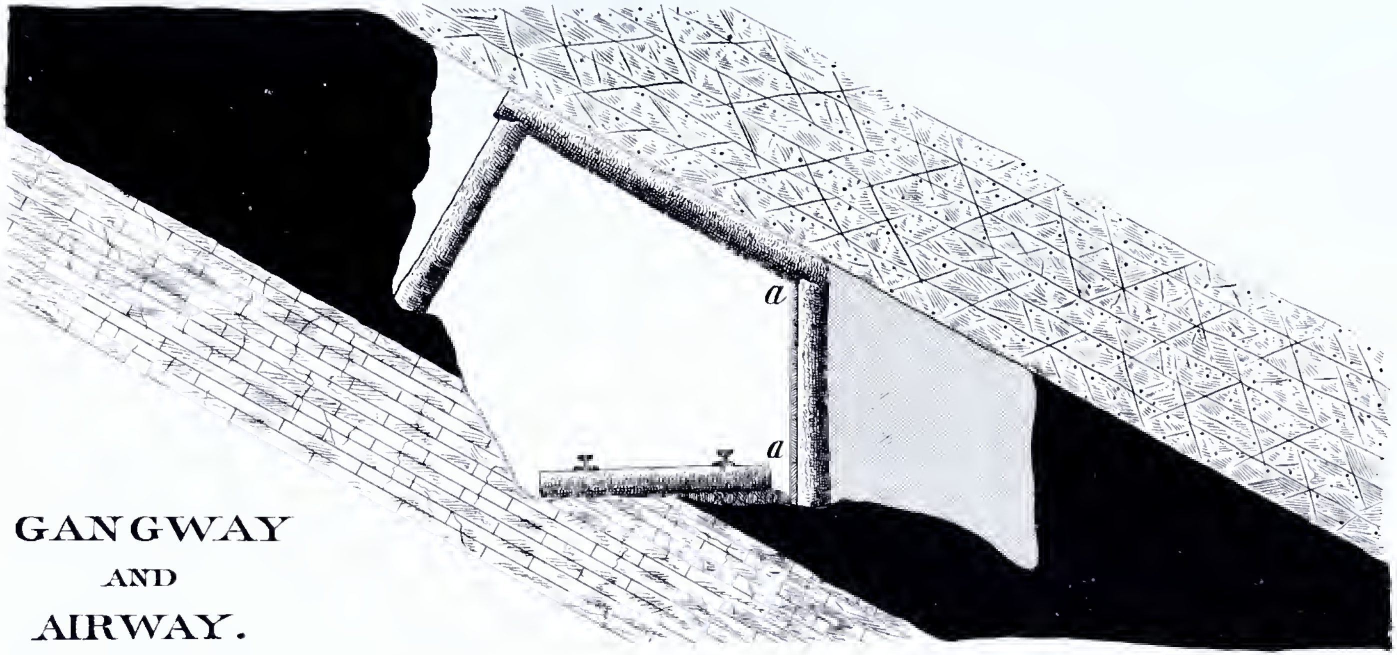
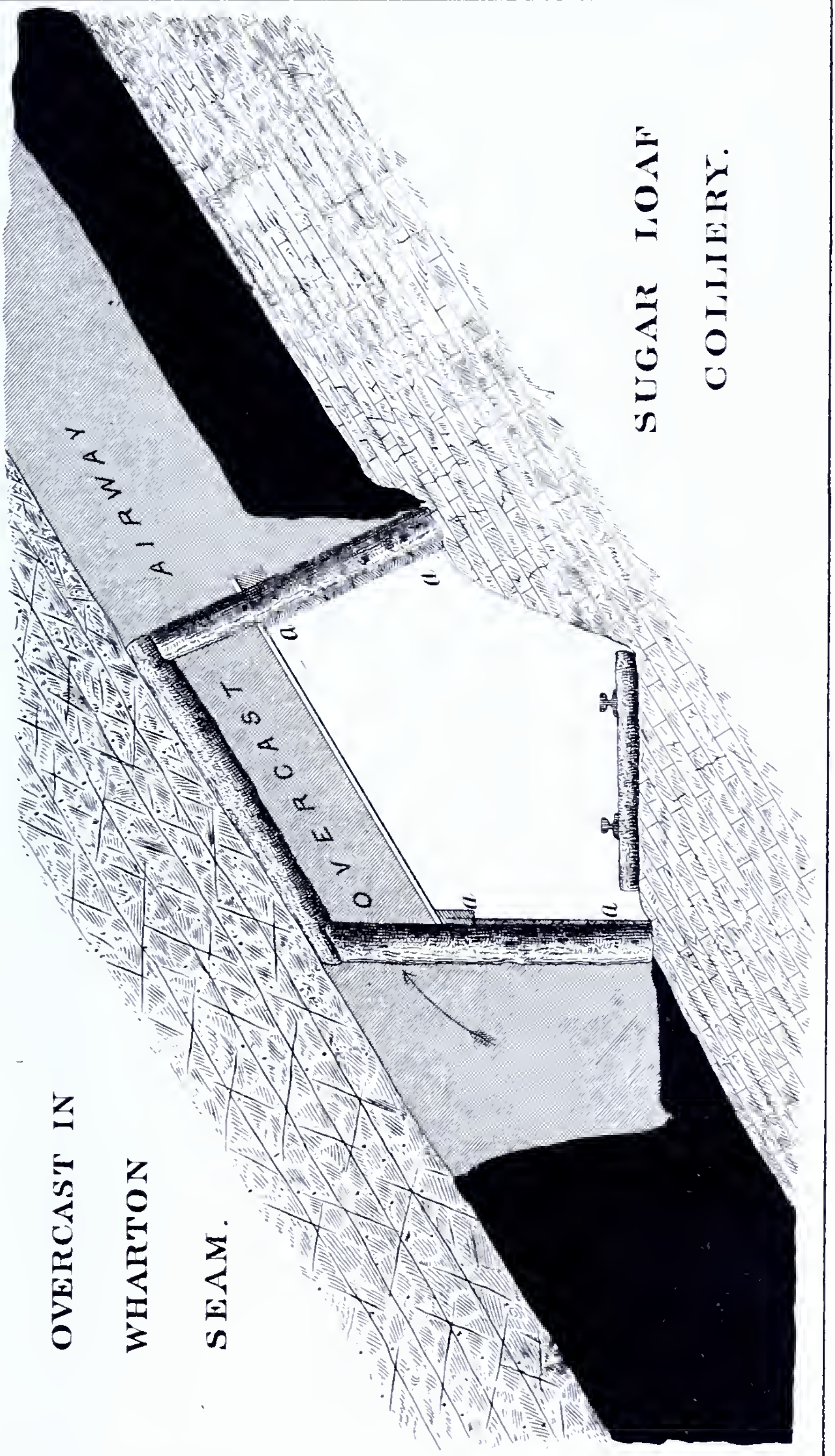


Fig. 51



GANGWAY
AND
AIRWAY.

WHARTON SEAM, HAZLETON.



OVERCAST IN WHARTON SEAM.

SUGAR LOAF COLLIERY.

the tunnel, and along the gangway (through a continuation of the same airway) to the face of the heading, thence up through the breasts and down to the gangway, thence around the inside mouth of the tunnel, through an airway cut in the upper benches, thence through an overcast across the gangway into a bratticed airway, thence to the face of the heading, out through the gangway and tunnel into the upcast.*

Page plate No. 38 shows the method of carrying the brattice along the gangway when the latter is timbered. The shaded portion represents the airway, and the brattice is shown at *a a*.

Overcasts.—The overcast crossing the gangway in Fig. 51 is shown in cross-section by Page plate No. 39. A brattice is built on each side of the gangway at *a a* and *a a*, and the overcast is built on stringers spiked to the mine timber. To insure sufficient headway the mine timbers (legs) are made longer, and the gangway is driven higher here than at other points.

Overcasts are built of timber in non-fiery mines, but at collieries making a large amount of gas masonry overcasts are preferred by many engineers. However, as frame overcasts can be more quickly and cheaply built than brick or stone arches, the latter are not as generally used as we should think advisable. It is evident that at fiery mines the best plan is to have all overcasts (when practicable) as independent airways driven in solid coal.

The method of ventilating collieries making large quantities of gas is generally such that overcasts are not often necessary, when we begin to mine our thinner seams, we shall doubtless be compelled to have many such passages, and to build them of brick, masonry, or iron. This is the present English practice.

Regulators.—A regulator is simply an opening made in a brattice, stopping or door, or on one side of a door, to allow a portion of the ventilating current to pass through to ventilate the passages beyond; it is provided with a sliding door, so that the amount of air passing into the split can be readily regulated. This door should be provided with a padlock or some other form of fastening, to secure it in any position the mine boss may think necessary.

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* Shown as a slope in the illustration.

Brattice cloth is used largely in place of doors and regulators to throw a portion of the ventilating current up through each breast. It is hung across the gangway from the mine timbers, reaching down to within one, two, three or four feet of the road-bed. It is also largely used in place of a brattice for throwing the ventilating current into the face of a breast or heading.

The law requires (Sec. 7) "not less than fifty-five cubic feet per second of pure air, or thirty-three hundred feet per minute for every fifty men at work in such mine, and as much more as circumstances may require, which shall be circulated through the face of each and every working place throughout the entire mine, to dilute and render harmless and expel therefrom the noxious, poisonous gases to such an extent that the entire mine shall be in a fit state for men to work therein, and be free from danger to the health and lives of the men . . . . and all workings shall be kept clear of standing gas ; . . . . . and there shall be an intake airway of not less than twenty square feet area, and the return airway shall not be less than twenty-five square feet."

With the exception of a few shallow non-fiery mines depending upon natural ventilation, the amount of air circulated is rarely less than four or five times the amount specified in the act. The operative clause of the act is that following the specification of thirty-three hundred feet, &c., viz: "and as much more as circumstances require."

Even when there is no danger from explosive gases a larger amount than this is necessary to quickly clear the atmosphere after blasting. Thirty-three hundred feet circulating through an airway of twenty square feet,—the smallest area allowable,—requires a maximum velocity of but one hundred and sixty-five feet per minute.

In the Wilkes-Barre district from 200,000 to 250,000 cubic feet per minute is circulated through some of the most fiery collieries.

In the shallow basins of the East Lehigh district many mines are entirely dependent upon the natural ventilation. When the workings are shallow, the amount of gas insignificant, and the mine has several openings at the surface, the natural ventilation is generally sufficient, but in summer it is often very sluggish, and a fan or furnace may be absolutely necessary in extremely hot weather.

Safety lamps.

The use of safty lamps is generally restricted to the work necessary in opening new workings. It has been repeatedly shown that none of the so-called safety lamps insure safety. They may cause an explosion when the ventilating current has a velocity exceeding four or five hundred feet a minute, they give a very poor light, and it is difficult if not impossible to prevent the miners from opening them.

The feeling of a majority of careful, well-informed miners is well indicated by the following remark made in my presence by an old miner: "God save the man when the gas will fire inside his (safety) lamp, for he'll never get out to tell it."

In driving gangways, airways, shutes and cross-headings to open new workings in gaseous coals, safety lamps are very generally used, also in inspecting working places where accumulations of gas may be expected, but there are few mining engineers willing to adopt them for mining.

Naked lights are almost universally used for mining, and to insure safety, it is therefore necessary to pass a sufficient ventilating current to dilute the gas and render it harmless.

This is generally very thoroughly accomplished throughout the region, and with proper care on the part of the miners and other inside employés explosions would be very rare.

Explosions now generally occur, not because the ventilating current is insufficient, but because it is not properly thrown up into the face and top of each working place where explosive accumulations of gas may occur.

It is entirely unnecessary to introduce into this report any information concerning the construction and relative merits of different forms of safety lamps, or to discuss the conditions under which they afford a certain degree of protection, as the literature on these subjects is already quite voluminous.

It will, therefore, be sufficient to state that nearly all colliery superintendents and engineers now unite in the belief that safety lamps should never be used for mining,—in other words, where the ventilation is insufficient to make the use of naked lights perfectly safe, mining should be suspended,—and that they should only be used in development work, (gangway driving, etc.,) and for examining the workings for gas, and erecting brattices, etc., to drive out any accumulation of gas that may be found in the breasts.

The use of luminous paint has been suggested as a substitute for safety lamps in gaseous workings and headings, but I do not know whether it has been found suitable.

Steam-jet ventilation.

Ventilating by means of steam jets turned into the upcast was but a few years since a very common practice, especially in the eastern portion of the Wyoming district.

I do not know that any collieries are so ventilated at present.

In shaft and slope sinking and during the opening of new workings, this method of securing a ventilating current is very often used with good results.

Furnace ventilation.

Prior to the passage of the Mines Ventilation act this was the principal ventilating method, but it has since been almost entirely superceded by the use of fans.

The law forbids the use of furnaces for ventilating when the "breaker and schute buildings are built directly over and covering the top of the shaft," (upcast,)—some engineers think the law should absolutely prohibit furnaces underground.

Only a few furnaces are now in use, most of which are located in the western part of the Wyoming basin.

It will not be necessary to enumerate the reasons why fans are now so generally preferred to furnaces, as these are obvious.

It has repeatedly been shown that for shallow workings a properly constructed fan will give a higher ventilating duty (water gauge or pressure × amount of air per minute) than a furnace, but that for very deep workings the furnace will give the best results. Few if any of our anthracite collieries have yet reached this depth, and when they do it will be found that the objections to furnace ventilation will be even greater than at present.

Fan Ventilation.

Fans of several different forms are used; they may be divided into two principal classes, viz:

1. Open running.
2. Closed periphery.

Open running fans (open around the whole circumference) have been largely used throughout the region, but are now generally replaced by closed fans embodying the essential features of the Guibal.

Some companies and individual operators still adhere to the open running fan. Thus all the fans (with one exception) in use at the collieries of the Delaware, Lackawanna and Western company are of this type.

It is conceded by a very large majority of mining engineers that open running fans necessarily waste a large amount of power by permitting a reëntry of air behind each vane, and also from exposure to high winds.

Many mine bosses are still imbued with the idea that open running fans must give the best results because they allow the air to escape freely from all parts of the circumference.

There are some details in the construction of the closed fans of Guibal type, now commonly used, which demand attention. I will note them in the following order:

1st. The spiral casing, shroud, or housing.
2nd. The inclination of the blades, paddles, wings, or vanes.
3rd. Curvature of the blades.
4th. Width of blades.
5th. The shutter.
6th. The expanding chimney.

Perhaps the most important and distinguishing feature of the Guibal fan is the spiral or circular and spiral housing. In this the fan differs from all others, (except the Schiele,) for nearly all inventors patenting improvements on the primitive radial winged fan, specifically claim the free exit of air from all points on the periphery, whereas the Guibal delivers all its air through one orifice.

Great differences exist in the form of casing now used, some engineers preferring the shroud spiral throughout, others preferring it partly circular and partly spiral. Three forms are in common use: 1. Complete spiral, commencing at the throat (discharge) and expanding regularly throughout; 2. Upper half circular, spiral commencing 180° (opposite) from the throat; 3. Three fourths (or thereabouts) circular, spiral commencing opposite the lowest part of the fan, about 270° from the throat.

From the results obtained with several large fans with various forms of shrouding, I should judge that it is not necessary to extend the spiral around the whole circumference, that one half or more, 180° to 220°, may with advantage be made circular and the angle subtended by the spiral need not be more than 130° to 160° to give excellent results.

When the spiral is too short the discharge "jumps" from one segment to that next it, occasioning a decided jar. If this is alone considered it is evidently best to use a complete spiral, but a certain loss in the velocity of discharge occurs in fans so constructed (from friction, reëntries, etc.).

For this reason it seems best to construct the casing partly circular and partly spiral, the length of spiral being just sufficient to prevent the discharge from perceptible "jumping" from one segment to that next it.

The blades are made straight or radial, inclined backwards, curved backwards, or inclined backwards with the tips curved forward.

Under the ordinary requirements of mine ventilation the angle of inclination of the blades is, within certain limits, probably not a matter of much importance, but until a series of careful experiments are made with direct reference to this subject, it will not be possible to arrive at any satisfactory conclusions. A spiral blade forming equal angles with radii drawn to any point between its outer and inner periphery has been found to give good results, and this is probably the best form for small ventilators running at very high speed under a heavy water-gauge. In ventilators used simply for displacing large volumes of air under a merely nominal water-gauge, the inclination of the vanes is doubtless an important factor in the amount of work done, and under such conditions, especially in open running fans, spiral blades set backward at a very considerable angle to the radius will probably give the best results; but as, in the use to which mine ventilators are put, a water-gauge of from one to three inches is usually developed, and as the *radial velocity of discharge* at the periphery rarely exceeds 400 to 600 feet per minute, the amount of angular deflection of the blades is probably of not much importance.

While a majority of the engineers believe that the vanes should be curved, or inclined backwards from the radius, the best angle of inclination is still a moot point.

When six blades are used they are generally built on a hexagonal spider and the angle is 60°; with eight wings a square or octagonal spider is used and the angle is 45°, and with ten blades a pentagonal spider forms the frame, and the vanes are set at an angle of 36°.

Curved blades are adopted by comparatively few engineers, probably because they consider them of no special advantage, and because the cost of construction (to secure rigidity) is slightly greater with curved than with flat blades.

Considering the fan as a centrifugal air-pump, blades of spiral form are theoretically the best; but when we consider that the only office of the blades is to *impart a circular motion to the air*, that they are not intended to *throw the* *air off*,—the centrifugal force developed does this,—the form of the blades does not seem to be of much importance.

It is a common practice with many engineers to use flat blades with the *tips curved forwards* so that they are perpendicular to the periphery, with the object of securing a tangential delivery of the air.

I am inclined to the opinion that this is rather detrimental than otherwise, and that if the vanes are curved at all, the curvature should be backwards. The form of the shroud effectually governs the direction in which the air is delivered without reference to the form of the vanes.

A width equal to one-third (or a little less) of the diameter fairly represents the average proportions now commonly used. That this width is far greater than that necessary to pass the required amount of air through fans of large dimensions is almost self-evident.

Experiments have shown that when used for simply displacing air, a fan of ordinary construction will pass an enormous quantity with a very small consumption of power,—frequently eight or ten times the quantity that can be circulated through the mine,—hence the capacity of the fans for passing air is generally largely in excess of the amount of air they can draw through the mine.

It would, therefore, be possible in many cases to obtain as thorough ventilation with fans one half or one third the width now generally adopted, but it is questionable whether this reduction in width could be effected without impairing the stability and rigidity of large fans.

For small fans running at high velocity a width of one third the diameter or more is doubtless necessary to prevent the fan resistance becoming too great.

Comparatively few fans are now built with an adjustable shutter. It is generally assumed that the dimensions of the orifice of discharge at the throat should bear a definite relation to the capacity of the fan and its speed of revolution (speed of periphery of blades) and that the spiral should be designed accordingly. To insure a certain margin for larger quantities of air that might become necessary under changed conditions, it is generally considered best to provide a certain excess in the width of the outlet.

Instead of providing a shutter, the orifice of outlet can be decreased by adding boards, planks, or plate-iron to extend the casing downward—in other words, to lower the position of the throat.

The width of the orifice of discharge may be calculated thus: making,

V=volume of air discharged per minute.
r=radius of fan.
w=width of fan.
n=number of revolutions per minute.
a=empirical friction constant=.75 to .90.
b=marginal excess (for altered conditions)=1.10 to 1.20
x=required width of orifice.

$$x = \frac{V}{2\pi r w n} \cdot \frac{b}{a} \qquad \text{(A.)}$$

The best position of the shutter can only be accurately determined by experiment, because we cannot determine in advance the exact value of a.

The factor a varies with the length of spiral, the leakage, and the velocity of revolution. It represents the velocity of the air within the spiral as compared with the velocity of the tips of the vanes, and may be assumed at .75 to .90.

The position of the shutter may be calculated thus:

The minimum width of orifice with shutter closed (position 0) is

$$x' = x - \frac{c}{d} x = \left(1 - \frac{c}{d}\right) x \qquad \text{(B)}$$

where x'=minimum orifice.
c=angle subtended by whole shutter.
d=angle subtended by spiral portion of casing.
x=maximum orifice (shutter in position 1.)

For any other volume of discharge at any other velocity we have

$$x'' = \frac{V''}{2\pi r n'' a} \qquad \text{(C)}$$

This orifice can be obtained by regulating the position of the shutter according to the formula,

$$p = \frac{x'' - x'}{x - x'} \qquad (D)$$

where p=required position of shutter, 0 being taken as the lowest and unity (1) as the highest position.

The proper adjustment of the shutter to the ever-varying requirements of mine ventilation cannot be intrusted to the mine boss,—it demands the personal attention of the mining engineer, but he can rarely find time to give to this work. We consequently find many fans provided with shutters that are never used, and many of the largest and best fans have recently been constructed without them.

That this is a retrograde step must be admitted; but at the mines where a few tons, more or less, of coal per month is a small matter, the saving of a few horse powers in driving the fan is not of much importance. The work of adjusting the shutter might be simplified by finding the value of a and calculating a table (by formula given) showing its position for revolutions varying from the maximum to the minimum number per minute, and for volumes of air varying between the extreme limits. With such a table posted up in the fan house, the mine boss could instantly adjust the shutter to its proper position. This table might be simpified by substituting for the volumes of air, a column showing the velocity of the air current at some convenient point in the main upcast airway.

The theory of the shutter may be briefly stated: The air is delivered at the throat at a velocity nearly equal to the tips of the vanes (if the spiral is properly proportioned .75 to .90, depending on the length of spiral, etc.,) and a certain known volume of air is delivered by the fan every minute. If the outlet is too small the air does not find free exit, if too large reëntries of air occur behind each blade, but the shutter provides a means of regulating the size of the orifice so that it may be just sufficiently large to give free exit to the required quantity.

A ready means of approximately fixing the shutter is to open it full width and then to very slowly close it until the throbbing stops ; but this will not always answer, because if the spiral is too short the fan will throb in any position of the shutter, not from reëntries, but from the discharge jumping from segment to segment.

The action of the expanding chimney is explained by nearly all engineers and manufacturers who have written upon the subject, by the assumption that the column of air thrown out through the throat diverges to fill the chimney, and thus meets the atmosphere at a reduced velocity, entailing less loss of power than if projected against the atmosphere at the speed at which it leaves the fan blades.

This explanation does not seem to me to be substantiated by the practical working of any of the fans I have observed I can perceive no reason why the column should expand to fill the chimney.*

When a column of air is projected through an orifice into an expanding outlet the momentum of the air carries it directly forwards (upwards), and it does not diverge to fill the outlet unless this latter is very long.

This can be seen at almost every fan in operation, especially when the air is moist and a cloud of condensed vapor marks the course of the air. The air column rises independently of the flaring sides of the chimney, and *downward reëntries* of air occur on one and often on several sides of chimney.

But the advantage gained from the addition of the flared chimney may be readily explained by the well-known reduction in loss of head resulting from the use of a divergent ajutage or mouth-piece.

This has been amply demonstrated by Weisbach, Venturi and Eytelwein in experimenting on the flow of water through ajutages of varying shapes, and it is this principle which is utilized in the Guibal *evasée* chimney But it may be thought that this principle is not applicable because the conditions are not identical ; for

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* Except that it is slightly condensed by the centrifugal force—a condensation so slight that we can ignore it.

1st. We are here dealing with air, a compressible body which immediately expands to its normal volume at the atmospheric pressure.

2d. That the fan delivers air, not strictly speaking, through an orifice, but from a rectangular passage that presents no appreciable obstruction to the free exit of the air (as the orifice through which water discharge experiments are made.)

3d. Reëntries of air usually occur on at least one side of the fan.

If the *evasée* chimney accomplished all it is designed to accomplish the Guibal fan should certainly give a much greater useful effect than has yet been obtained from it, but while its action is doubtless imperfect it is certainly a valuable addition to any fan.

Experiments made on many different forms of Guibal fans, principally by English and French mining engineers, show that that the average useful effect obtained in the air is not more than 60 per cent. of the power indicated in the steam cylinder. If we allow 20 per cent. for loss from friction, (with direct-acting engine,) we have still a loss of 20 per cent. due to some imperfection in the fan.

We can attribute a small portion of this loss to leakage, but it is evident that the greater part of the loss is due to some other imperfections. When the fan is used as a "blower" the useful effect should be correspondingly greater, because the principle of the *evasée* chimney (reversed) can be successfully utilized.

A series of experiments carefully made by Messrs. Gille and Franeau,—a translation of which may be found in vol. xvi, 1866–7 of the Transactions of the North of England Institute of Mining Engineers,—has most clearly demonstrated a marked increase in the useful effect due to the addition of the spiral casing, the proper adjustment of the shutter, and the expanding chimney. These experiments were made at the collieries of Crachet and Picquery on a fan with blades inclined backwards from the radius at an angle of forty-five degrees.

Fans of very large diameter are now coming into use at the fiery collieries of the Wyoming district, but even these are small compared to some in use at English collieries.

As the water-guage produced by a fan is dependent upon the velocity (squared) of the tips of the vanes, as the quantity of air obtained is dependent upon the water-guage, and as comparatively small fans can be run at as high a peripheral speed as large fans, many engineers are opposed to the adoption of very large fans, believing that smaller fans will give the same results at less cost.

A peripheral speed of about three thousand feet per minute (60 revolutions for a 16 foot fan) is generally considered to be about as high a speed as it is well to adopt under ordinary circumstances, but the fans are commonly constructed strong enough to be run up to nearly or quite double this speed in emergencies.

Very large fans are run relatively somewhat slower than small fans.

Atlas sheet No. XIII shows the style of fan now used by the Lehigh and Wilkes-Barre Coal company. It is thirty-five feet in diameter, with masonry pillars, brick side walls, and plate-iron casing and chimney, so as to be practically fire-proof. It is driven by a direct-acting horizontal engine. Upright engines are largely used for fans of moderate size (16 to 20 feet), but they are disliked by the engineers in charge of them, because it is impossible to keep them clean. Small fans are frequently run by belting, but nearly all mine superintendents prefer direct-acting, to geared engines of any type for this purpose.

Too much importance cannot be placed upon the selection of a fan engine for a large fiery colliery ; a break down may at any time cause the loss of many lives and great damage to the mine, and it is, therefore, of great importance to have an engine that will run regularly without close watching and with little risk of breakage, and that can be quickly repaired or replaced in case of serious damage.

There are as yet comparatively few fans more than twenty feet in diameter, a very large number of twenty-foot fans , and a still larger number of sixteen-foot fans are in use ; there are very few now used less than ten feet in diameter, but quite a large number of old twelve and fourteen feet fans are in use.

Many of these latter, as well as some of the sixteen-foot fans, are old open periphery radial-winged fans, to which have been added the spiral casing and flaring chimney of the Guibal type.

The Philadelphia and Reading Coal and Iron company have recently built their fan casings of wrought-iron, and the Hazleton and other shops have turned out similar work, but the number of fire-proof casings is still comparatively small.

However, the feeling among the mining superintendents and engineers on this subject is such that in the future there will probably be few if any new fans erected at fiery collieries with wooden casings.

Water-gauge.

The water-gauge developed by a fan is theoretically (approximately) proportional to the square of the number of revolutions and also to the square of the radius or diameter—in other words to the square of the velocity of the tips of the vanes, (blades). Efforts to obtain an observed water-gauge that shall agree with that obtained by calculation almost invariably fail, and it is doubtful whether we shall ever obtain a formula practically useful as a check on water-gauge readings.

As at present used the water-gauge is rarely, if ever, a reliable instrument. It is commonly placed in a hole in a stopping between the in-take and return airway, either of the mine or of any split, and the hole luted with clay or otherwise made nearly air-tight.

Sometimes the stopping in which the water-gauge is fastened borders directly upon the airway, and in that case a current of air is flowing past, or directly against, the mouth of the gauge, increasing or diminishing the reading by the suction or pressure due to the velocity. With very high velocities this error may reach serious proportions. If the stopping is located midway in a heading, or at a considerable distance from the airway, the current of air flowing past it may increase the reading.*

It is frequently claimed that water-gauges permanently fixed at certain points might be of value in determining when a roof-fall or other obstruction, or an open door, had impaired the ventilation. While this is doubtless true, it is also true that the miners usually discover at once from the state of the ventilating current when anything is wrong, but unfortunately not always soon enough to prevent an explosion.

The water-gauge is principally useful in determining from time to time that the condition of the airways is properly maintained to get the best results from the fan.

The water-gauge (or in other words the ventilating pressure) rarely exceeds two inches. At a few collieries it runs up to three inches, but as a rule it is from half an inch to one inch and a half. The secret of the large quantity of air passed through some of the mines of the Wilkes-Barre district—two hundred thousand feet and more per minute—is in the small water-gauge obtained by large airways, and the method now adopted at all collieries (when possible) of dividing the ventilating current into as many splits as possible.

In Mr. J. J. Atkinson's essay on the "Friction of Air in Mines" we find a clear exposition of this subject, which may be summed up in these few words: To reduce the friction, and consequently to increase the quantity of air with a minimum expenditure of power, enlarge the airways and split the current into several sub-divisions.

The value of his teachings is probably nowhere more amply demonstrated than in our anthracite mines, notably those of the Wilkes-Barre district, where the ventilating currents are probably larger, with lower water-gauges than in any other mining district in the world.

At several large collieries the water-gauge is little more than half an inch, and there are now comparatively few large collieries in the gaseous portion of the Wilkes-Barre field at which the water-gauge is more than one inch or an inch and a quarter.

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* As in a water syphon that will suck in air or fluid through any opening.

At collieries making only a small amount of gas the ventilating pressure is frequently from one and a half to two and a half inches of water-gauge.

Remarkably high water-gauges with an enormous volume of air have lately been reported from the double fan erected at the Baltimore Tunnel mines of the Delaware and Hudson company. This apparatus consists simply of two seventeen-foot fans of Guibal type with complete spiral casing, eight blades bent backwards from the radius and mounted on double spiders placed in the center of each fan. The two fans are provided with separate casings, are mounted on a single shaft about fourteen feet apart from center to center, and are driven by belting (6 and 12 foot pulleys) by a pair of 16″ × 30″ engines.

The policy of building two fans on one shaft or of placing two fans over one upcast is not indorsed by the mine superintendents and engineers throughout the region. Two fans so built must give practically about the same effect obtained from one fan twice as wide as a single fan*, and it has already been shown that the width is a factor of small importance. Fans built wide enough to insure a good degree of stability, are always wide enough to readily pass any current that can ordinarily be drawn from a mine.

The results claimed for this fan, and for other double ventilators, are viewed with distrust by nearly all anthracite colliery superintendents and engineers.

At a few collieries the experiment of placing two fans on one upcast has been tried,—the results have been found to be exactly what we should naturally expect,—no appreciable benefit.

As the quantity of air circulated through a mine is directly governed by the ventilating pressure (water-guage) and as this depends upon the speed of the tips of the vanes, and not upon the number of fans or width of the blades, it is evident that with two fans of equal size running at equal

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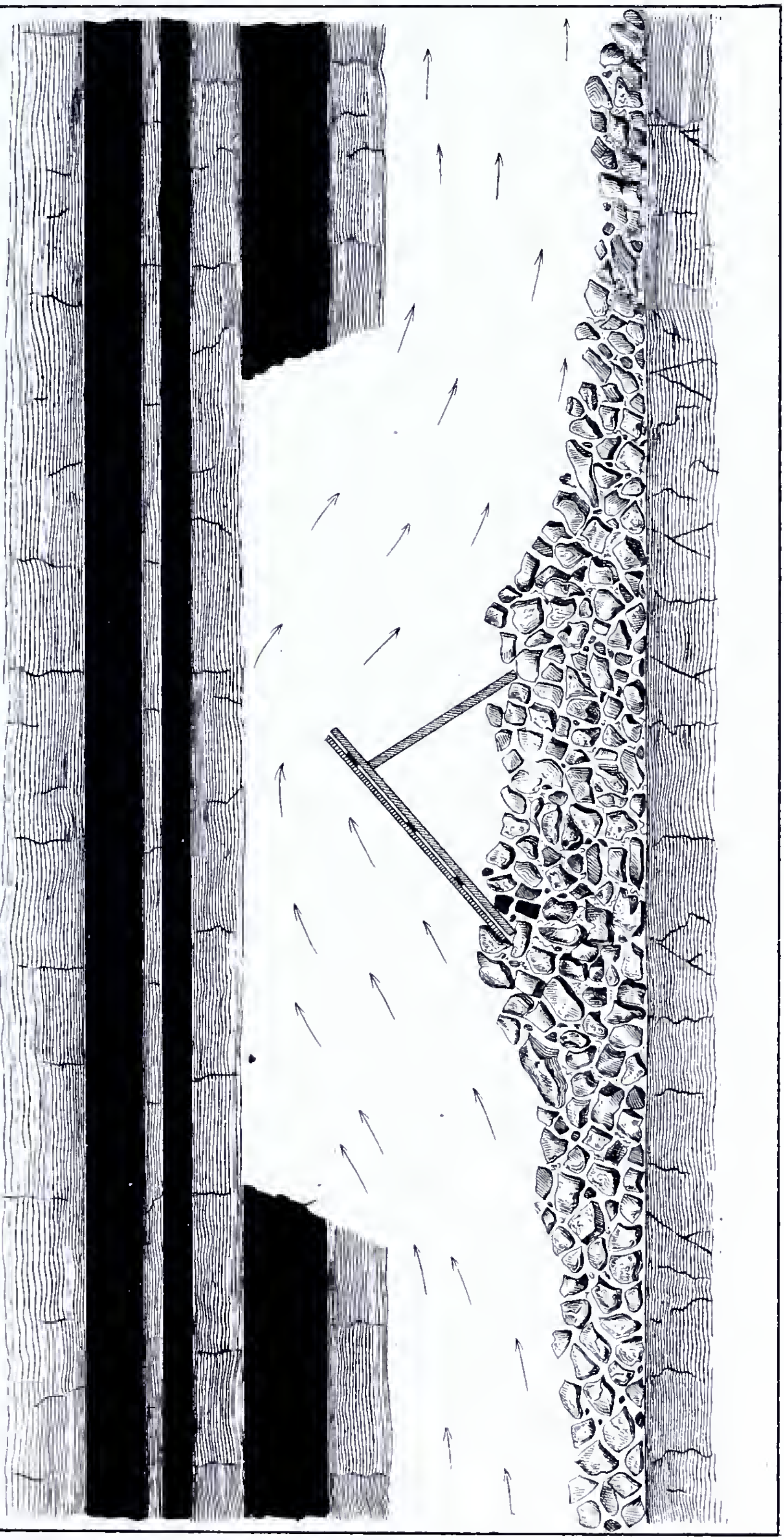
* Except that the *fan resistance*, which is always very small in well built fans, is somewhat less.

SLANT BATTERY.



speeds, the water-guage (barring fan resistance) will not be appreciably greater than with one fan.

At a few collieries where a very high water-guage is necessary, the air is passed through two fans, the exhaust air from one being turned into the in-take of the second, thus effecting double rarefication and obtaining a water guage (nearly) double that produced by a single fan. When this arrangement is adopted, either by placing one fan inside the mine or having both at the surface, it is not essential that they shall either be both of the same size, or run at equal velocities.

When the size of the fans and the speeds of revolution are unequal, the resulting water-guage is approximately equal to the sum of the water-guages produced by both fans.

The law provides (Sec. 9) that "every mine having explosive gas . . . shall be divided into two, four, or more panels or districts, each ventilated by a separate split or current of air, and fifty persons shall be the greatest number that shall work in any one panel or district at the same time. . . ."

In lift mining it is customary to divide each lift into two splits, one on each side of the slope; with two lifts working there will be four splits, and with three lifts six splits. Counter-gangway workings are usually ventilated by a split from the main gangway below, or by an independent split from the main in-take. (See Page plate No. 14.)

The intake at shaft workings is usually split into two main sub-divisions near the foot of the shaft, running off along the gangways opened in opposite directions from the foot of the shaft. Separate splits are taken off for each counter-gangway, and for each panel of workings operated by a plane or by an inside slope.

When there are several in-takes, as is frequently the case in shallow workings and at slope collieries where the workings are holed through to the outcrop, the separation of the current into splits is readily accomplished.

An excellent example of workings of this class is shown by Atlas Sheet No. XIV.

Ventilation of breasts.

The method of ventilating breasts is sufficiently well shown by Page plates 18, 19, and 20, and by the illustrations composing Atlas Sheets XXI, XXII, and XXIII.

The method shown by Fig. 52 has been used by the Lehigh Coal and Navigation company for single shute breasts in thick steep-pitching coals. It differs from those shown by the illustrations above cited, but has been found unsuitable for fiery workings because any obstruction at the face, or in the juggler manways above the last cross-hole, impairs the ventilation of all the workings beyond the breast in which the obstruction is located.

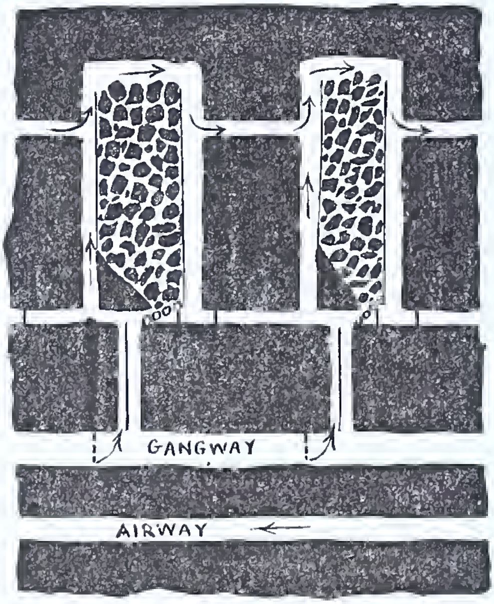


Fig. 52.

For this reason it has been partially replaced by the plan shown by Figs. 53 and 54, in which each breast is ventilated by a separate split of air taken off from the gangway through the travelling-way in the main shute, and returned through a cross-hole driven through from the first or main heading into the return airway.

In Fig. 53 (and also in Fig. 52) the airway is represented as being below the gangway. It was drawn in this position to simplify the illustration. In the cross-section shown by Fig. 54 it is shown in place,—*c* being the return airway; *m* the cross-hole driven through into the heading *a* near the foot of the left-hand juggler manway (airway).

It will readily be seen that when this plan is adopted an obstruction in a breast does not affect the ventilation of any other workings.

This plan is more expensive than many others, and in soft coal is objectionable because the cross-hole (*m*) weakens the stump,—but in fiery workings its advantages will probably prevail over any objections that may be urged against its adoption.

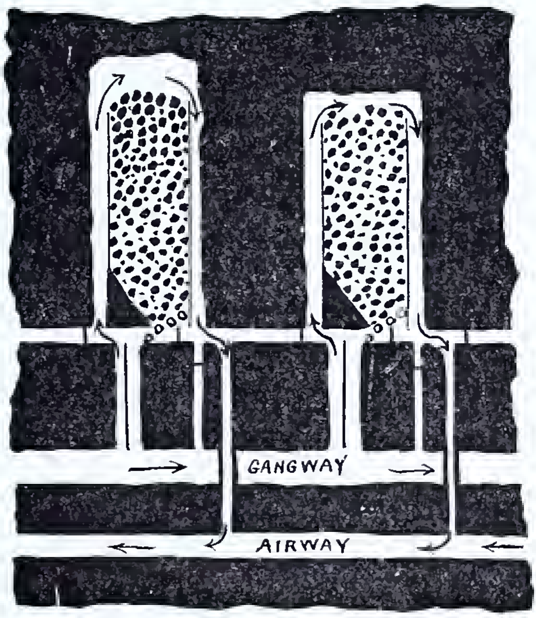


Fig. 53.

It will be understood without further explanation that in ventilating all steep-pitching breasts the air is either held in the breasts by air-tight stoppings or check batteries, by stoppings with regulators to allow only a small part of the air current to pass directly into each breast, or by door or brattice-cloth ("sheets") hung on the gangway or in the main shutes, and that all other passages (slant shutes, manways, etc.) are closed by stoppings or trap-doors. (See Page plates Nos. 18, 19, and 20.)

When the air is coursed back through the breasts and cross-headings, which is the most common plan, the shutes and travelling-ways are all closed, except at the last breasts in the panel, and the air is coursed back through the breasts into the main upcast, but as none of these stoppings can be made air-tight, a small split of air enters each breast. To secure the best results by decreasing the friction of the air, the cross-headings connecting breasts nearest the main upcast should, therefore, be relatively larger than those near the limit of the workings, but when the air is returned through an independent air-course the reverse is evidently true.

When this latter plan is adopted the air is forced into the workings (Fig. 52) by brattice sheets (or doors and regulators) hung across the gangway, or by regulators in the travelling-way which is bratticed off on one side of the main shute.

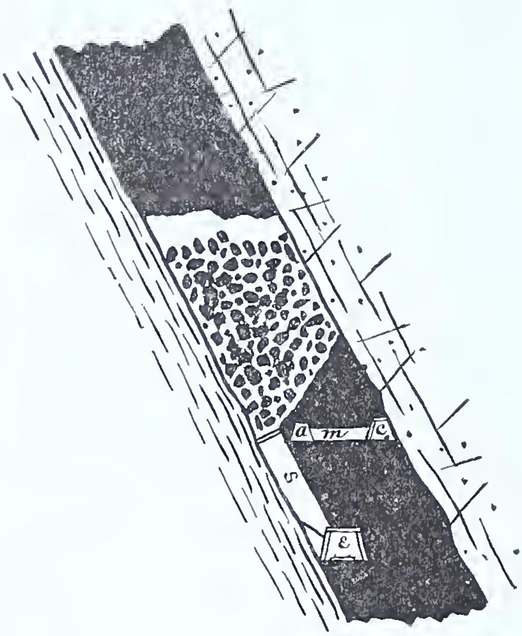


Fig. 54.

In flat or slightly-inclined gaseous workings, the shute* opening into the breast is closed by a door opening near the gangway in wagon breasts, or in buggy breasts, by a brattice with a draw-hole closed with brattice cloth, under which the coal is dumped to fall on the platform below.

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* An improper use of the word; the English term "bolt-hole" seems peculiarly appropriate,—we might call such passageways *breast entries.*

When the bed is thick or steeply inclined, accumulations of gas near the roof, or in holes in the top caused by roof-falls, are prevented or dispelled by deflecting the current by brattice or brattice cloth carried up from the last cross-hole or from the juggler manway which is within a few feet of the face.

Ventilation by blowing.

Ventilation is, with few exceptions, effected by suction, either by furnace or exhaust fan. At the collieries worked under the supervision of Col. D. P. Brown of Lost Creek, superintendent of the Lehigh Valley Coal company's collieries in that neighborhood, we find the ventilation effected by fans used as blowers. There can be no doubt that as large a ventilating current can be produced by a fan when so used as by suction ventilation ; and it seems probable that better results may be obtained by blowing than by suction, if we measure the value of these results by the number of cubic feet circulated per minute. The great loss of power caused by projecting the *exhaust* current of air against the atmosphere at high velocity, is avoided, and while a certain loss of power results from throwing the current into the mine at such a high velocity, some of the power so expended is utilized by imparting a certain momentum to the air in the downcast.

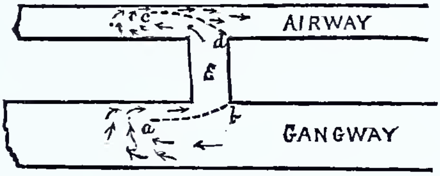

Fig. 55.

The special advantage claimed by Col. Brown for this method is that the headings of gangways and airways, etc., can be more thoroughly ventilated, and the cross-headings need not be so close together to insure thorough ventilation in gaseous workings.

Thus in Fig. 55 showing the ventilation of an airway and gangway heading by *suction*, the air current comes slowly up the gangway, turns around the brattice cloth hung at *a b* into the cross-heading *e*, and slips around the brattice cloth *c d* into the return airway. The brattice cloth *c d* must be hung in this position to allow coal from the airway to be dumped through *e* and loaded (under the brattice cloth *a b*) into wagons on the gangway; while the brattice *a b* is so hung to allow the wagon to pass in to the face.

Ventilation of the same headings by *blowing* is shown by Fig. 56.

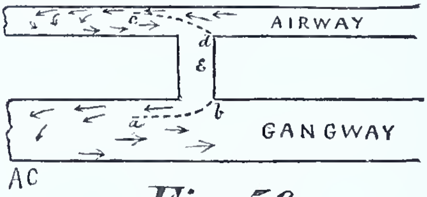

Fig. 56.

It will be observed that the ventilating current coming in through the airway, passing along behind the brattice, is confined to a narrow passage and acquires great momentum (velocity), which carries it well into the face; the same result is obtained on the gangway, both headings being thoroughly ventilated up to the face.

The chief disadvantage in ventilating by blowing is that the foul air, explosive gases, powder, smoke, and steam are carried *out through the gangway.* As the gangway is the main haulage road, this constitutes a serious objection to the use of blowing fans, and more especially in gaseous workings where safety lamps are required in driving headings and opening new workings, because the explosive gas (diluted, of course, with a large quantity of air, but which may become explosive) is blown out through the gangway and traveling-ways where the employés are using naked lamps. At the same time it is claimed that this system furnishes an additional safeguard against accident, as it absolutely requires that there shall be sufficient ventilation to make the gangways perfectly safe—and when the gangway air is safe, all of the working places (by this system) must necessarily be free from danger.*

It is, however, a very serious drawback to the proper working of any mine to have the mainways filled with foul air, clouded with powder smoke, and perhaps steam.

The style of fan preferred by Col. Brown is roughly shown by Fig. 57.† In some respects it resembles the Schiele fan, but is doubtless more efficient.

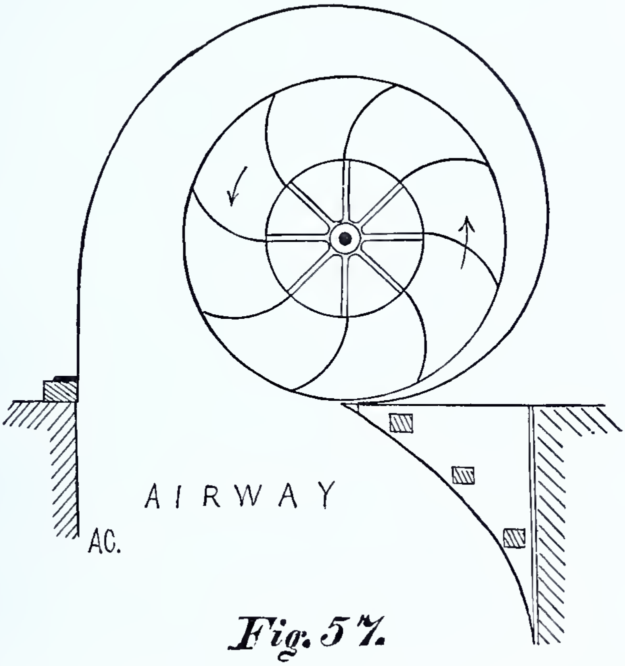

Fig. 57.

The fan has eight wings curved backwards from a point one-half the radius distant from the center. Diameter 16 feet, width 5 feet. The shroud or casing is spiral throughout.

The fan exhausts (blows) into an intake airway 10x12 feet.

Several methods of measuring the amount of air are in use. They all depend upon correctly ascertaining the velocity of the air current, which is then multiplied by the cross-sectional area of the passage to find the volume of air circulated per minute. The area of the airway (whether gangway, intake, upcast, or return airway) is obtained by careful measurement. A measurement made at one point is not sufficient to insure accuracy (except for anemometer measurement), and it is advisable to take the mean of measurements made at several points, the smaller measurements being taken in preference to the larger ones.

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*Except from accumulations near the roof, etc.

† The circle marking the outer periphery of the fan blades should be erased. The figure shows the fan in section, the inner circle representing the intake in the fan shroud. The outer circle is a mistake of the draughtsman.

Having determined the size of the airway, the velocity of the air is taken either with an anemometer or by one of the following methods:

1st. Noting the time powder smoke takes to travel a certain distance previously measured, or the distance it travels in a fixed time; or

2nd. Substituting for the gun-powder, ether, bisulphide of carbon or some volatile fluid or gas with strong odor;

3d. Noting the time, or distance traveled in a fixed time, by walking with a lamp at equal speed with the ventilating current, so that the flame remains vertical.

As the anemometer requires very much less time and trouble than any other method and does not require the services of an assistant, it is nearly always used,—but the above methods furnish a valuable means of checking the anemometer readings and of determining what corrections are necessary.

Several different styles of anemometers are now manufactured, (principally by English and French makers,) but the well-known Biram instrument is preferred.

Every anemometer is more or less liable to irregular working, either from the presence of dirt, gummy oil, undue friction, or from wear of its working parts, and for this reason it is considered advisable to make periodic tests to determine its friction constant.

The anemometer dials score the number of revolutions made by the wheel, assuming each revolution to represent one foot of distance; but as the inertia of the instrument does not permit it to start off at full speed, as there is also a certain resistance due to starting friction, and a certain resistance due to friction after the instrument has attained its full velocity, the record of the dials cannot be assumed to represent the actual velocity of the air.

It is evident that a certain number of revolutions should be added for the resistance due to inertia, a certain number for the resistance due to starting friction, and that allowance should be made for resistance due to running friction. As the instrument makers in constructing the instrument, usually endeavor to apply a correction for running friction, and as they often make this allowance too large, the actual correction to be made for running friction at small velocities is usually a *deduction* of from 1 to 4 *per cent.* for new instruments in perfect order; for old instruments the correction is smaller, and should also be smaller for high velocities, and for very high velocities in old instruments an addition may be needed amounting to one or two *per cent.*

It is evident that the allowance (addition) to be made for starting friction and inertia of the machine, is not the same for all velocities nor for observations extending over different periods of time; hence no rule can be laid down which will give absolutely correct results.

A method commonly used is to multiply the revolutions recorded by .97 and add forty or sixty feet for inertia and starting friction, according to the formula

$$v = .97\ \mathrm{R} + 60;\ \text{or}$$
$$v = .97\ \mathrm{R} + 40.$$

Still another plan is to adopt the following:

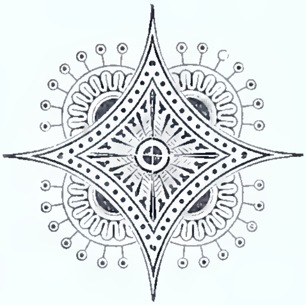
$$v = .97\ (\mathrm{R} + 60)\ \text{or}$$
$$v = .97\ (\mathrm{R} + 40).$$

A more correct plan is to adopt the following formula and to determine the values of a and $(b + c)$ experimentally for one, two, three, or five minute observations:

$$v = a\,\mathrm{R} + (b + c)$$

where $a =$ instrumental correction,
$b =$ correction for inertia,
$c =$ correction for starting friction,

and after finding the values of a and $(b + c)$ for several different values of R, to construct a table of corrected velocities.



CHAPTER XX.

Colliery Management.

Under this head a variety of subjects may properly be discussed that have not found a place in any of the preceding chapters.

Efficient colliery management not only decreases the risk of accident to miners, laborers, and to the mine itself, but must always result in effecting the most economical extraction and preparation of the coal.

A cursory examination of the ground upon which the surface plant is erected, will often furnish sufficient evidence as to the character of the management. At a poorly managed colliery we may commonly find old, half-worn mining tools, bolts, car axles, wheels, pumps, sheaves and all kinds of old iron work, damaged mine cars, etc., scattered around in confusion; this is presumptive evidence at least that the management is lax.

Good management requires that old supplies, iron work, etc., should be properly examined, and either stored away for future use, or condemned as worthless, and sold as old iron.

The condition of the road-beds of the surface plant, also generally furnish a tolerably reliable index to the condition of the roads underground.

In the anthracite mining regions, more especially in some parts of the Schuylkill and Lehigh districts, the means of obtaining a good supply of water for the boilers is now one of the most important subjects.

The water supply is commonly obtained by damming up small mountain streams, and conveying the water through pipes to the colliery.

This plan is largely used in the Wyoming, the Schuylkill, and parts of the Lehigh districts.

Where a large river is available, as in the Wyoming district from Pittston to Nanticoke, fair water may be had at all seasons, but in the districts depending upon small mountain streams, there is always great scarcity of water in dry seasons, and sometimes during very cold weather.

Under such circumstances there are but three remedies: *1.* To bring (pump*) water from larger streams located outside the coal-producing area, and therefore not contaminated by mine water; *2.* To purify the mine water, either by neutralizing or removing the destructive acid and salts with which such water is usually loaded; *3.* To sink deep artesian wells to obtain a supply from rocks underlying the coal measures.

The Lehigh Coal and Navigation company, the Lehigh Valley Coal company, and the Philadelphia and Reading Coal and Iron company, have all adopted the first plan, but as the available streams are few and small they only insure a supply for a small number of the working collieries, and in very dry weather many of the collieries must run on mine water or shut down.

An efficient means of purifying or neutralizing (or both) the acid mine water is therefore a matter of no small importance.

A large number of patented or secret compounds are offered in the market for preventing scale in boilers. Many of these answer very well for water in which lime is the principal impurity, but for mine water loaded with sulphate of lime and iron, and containing a large amount of free acid, they are generally useless, not because some of them do not contain the proper ingredients, but because they do not contain them in the proper proportions.

Tannin, saw-dust, chloride and carbonate of ammonia, soda, chloride of barium, wood pulp refuse, etc., have been used with varying success.

Chloride of barium and logwood or tannin form a mixture from which good results have been obtained. Col. Brown uses the following mixture:

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* Tank cars have been largely used for bringing water from a distance.

Chloride of barium,	1 part.
Logwood,	4 parts.

This mixture throws down the acid as sulphate of barium and converts the lime into a chloride, the iron being converted into a tannate. These compounds form a mud, which is blown out from time to time.

Chloride of barium is undoubtedly one of the best substances that can be used, but is objectionable on account of its cost.*

The so-called "feed-water heaters" are not successful in treating mine water unless sufficient lime or other material is first added to the water to neutralize or combine with the sulphuric acid to form a compound that will be precipitated at a certain temperature; hence they have not been adopted.

The plan lately adopted by the Philadelphia and Reading Coal and Iron company consists of adding a sufficient quantity of lime to the water to take up all the acid, mixing thoroughly in a tank (capacity about 100 bbls.), allowing the sediment to settle and drawing the water off into the feed-water tank. The quantity of lime necessary can be determined by analysis or by adding lime until the water shows no trace of acid when tested with litmus paper.

This process simply prevents the acid from attacking the boilers, and the water so treated will cause a heavy scale of lime. This can be almost entirely prevented by passing the water so treated through a feed-water heater to precipitate the lime salts.

The amount of sulphuric acid in mine water varies very largely. At some mines it has been found to reach 100 or even 200 grains per gallon. Such water will destroy iron with alarming rapidity, and cannot be used in boilers under any circumstances. Water containing only two or three grains (or less) to the gallon has been known to ruin boilers in a few months.

The practice of using condensed water from the exhaust does not seem to be advisable unless the lubricating oil (cylinder oil) is absolutely free from animal fats. The fatty acids form an insoluble lime soap, which coats the boiler at the water line, repelling the water and allowing the spots thus coated to become very hot,—so hot that oxidation is accelerated, the boiler weakened and possibly ruined.

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* About ten cents a pound.

If the water is to be used twice over, the expenditure of a few dollars more for a pure lubricant—even if the oil account is increased—is certainly a good investment.

The "anti-incrustator," prepared from wood pulp,* is said to consist of the ulmate and humate of soda, and it is claimed to act as a cheap and efficient neutrilizer of mine water.

Mr. Hugh Burgess, of the American Wood-Paper company, explains its action thus:

The free acid of the water is supposed to unite with the soda forming *sulphate of soda*, and leaving the humic and ulmic as free acids.

The sulphate of iron is converted into sulphate of soda, and humate and ulmate of iron.

The sulphate of lime into sulphate of soda, and humate and ulmate of lime.

The free humic and ulmic acids are said to be inert†, that is, that they will not attack iron or brass work; the other precipitates are in a finely divided state, and may be blown out as mud.

Mr. Burgess says: "I believe the scaling of steam boilers is closely connected with the electrical condition of each boiler. I have been surprised at the large amount of electrical current which passes, say from the steam drum to the bottom of the boiler (outside shells in both). I cannot help thinking but that this is solely a thermal current. I also think that the position of a steam boiler in relation to the magnetic meridian has much to do with the scaling . . I have been able to keep one set of steam boilers perfectly clean with Anti-incrustator, but have failed to do so with another set *fed with the same water*."

If, as is claimed for it, this Anti-Incrustator (the humate and ulmate of soda) is inert, containing no free alkali, it possesses important advantages over many of the compounds offered for this purpose.

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* Paper mill product.

† They are evidently nearly, if not quite, insoluble.

Some of these latter contain free alkalies which are very destructive to the brass work, etc., and others (those containing carbonated alkalies) throw down dense precipitates of iron almost as injurious as a hard scale.

The lime and feed-water heater treatment, the chloride of barium and logwood process, and the use of absolutely pure mineral lubricants, are the methods most favorably known among anthracite mining engineers for the neutralization of acid water, and prevention of scale.

Winding from slopes.

While at shaft collieries, and slope collieries using cages, only one car is raised at each winding, it is customary to wind two, three or more cars at once at slope collieries at which the cars run on the slope tracks, thus effecting a certain economy in the power expended.

Assuming the weight of the car (c) at one ton, the load of coal (l) at $2\frac{1}{2}$ tons, the weight of the full length of rope (r) at 2 tons, we have the weight (w) at starting from the bottom, (raising one car at each trip:)

$$w = (c + l + r) - c = l + r = 4\tfrac{1}{2} \text{ tons}$$

at middle of lift:

$$w = \left(c + l + \frac{r}{2}\right) - \left(c + \frac{r}{2}\right) = l = 2\tfrac{1}{2} \text{ tons}$$

and at the finish:

$$w = (c + l) - (c + r) = (l - r) = \tfrac{1}{2} \text{ ton.}$$

Winding three cars at once we would have ($r = 3\frac{1}{2}$ tons);

$$w = 3\,(c + l) + r - 3c = 3\,l + r = 11 \text{ tons}$$

at the middle of lift:

$$w = 3\,(c + l) + \frac{r}{2} - \left(3\,c + \frac{r}{2}\right) = 3\,l = 7\tfrac{1}{2} \text{ tons}$$

and at the finish:

$$w = 3\,(c + l) - (3\,c + r) = 3\,l - r = 4 \text{ tons.}$$

The advantages accruing from this plan are evident:

1st. A reduction to one-half or one-third in the speed of winding necessary when only one car is raised at each winding; 2d. Less variance between the load at starting and landing; 3d. The starting load is less compared to the amount of coal raised at each winding, thus permitting the use of gearing that will give better results throughout the winding. This is especially valuable in second-motion engines. Other advantages—coupling and uncoupling of cars, etc.—are sufficiently obvious.

When direct-acting engines are used, the number of cars raised at each winding is necessarily limited by the load which the engine can overcome at starting.

I have purposely omitted all mention of the system now used to some extent in foreign countries of counter-balancing the weight of the rope at shaft collieries, by attaching tail-ropes, or both ends of a single rope, beneath the winding cages.

While this plan of counter-balancing the weight of the rope, theoretically and practically, accomplishes this object perfectly, it has not been adopted in the anthracite regions.

Many objections have been advanced in opposition to this method; breakage of the tail-rope, swaying in the shaft, fouling in the (coiling) rope-pit, or wedging fast at the bottom sheave, straining and wrenching the cage, increased weight on drum and top sheaves, etc. But the chief objection to its use, and one that will probably always exclude it, arises from the fact that it greatly increases the weight on the rope near the rope-socket. This is almost universally conceded to be the weakest part of a winding rope, and to increase the strain at this point by the addition of from two to four tons weight, is a proposition that will not be favorably received by anthracite mining engineers.

At slopes and planes operated by a barney, a tail-rope is often necessary, but such ropes are usually much lighter than the main winding rope, and consequently act only as a partial counterpoise.

Inspection and repairs.

Probably the most important feature of efficient colliery management is a thorough and periodic inspection of the machinery and all structures forming the mining plant, as well as of the mine timbering, road-beds, workings, signaling apparatus, etc.

Before the general adoption of safety cages, the ropes, indicators, engine, and winding gear undoubtedly received more careful attention than is usually accorded them at present, and to this we may attribute the comparatively small number of accidents due to rope breakage.

At present the management at many collieries is surprisingly lax in this respect. At some collieries periodic inspection of the ropes is almost entirely unknown, a knowledge of their condition being obtained from time to time from casual reports of the engineer in charge of the winding engine, or from the mine boss.

And even when the rope is known to be in very poor condition, several days or even weeks are not infrequently allowed to pass before it is changed.

This delay is due sometimes to false economy, sometimes to indifference, and frequently to the fact that a new rope is not ordered from the manufacturer until it is actually needed, and then some time must elapse before it is received.

It is, perhaps, needless to urge the necessity of frequent and *periodic* inspection of the ropes, of keeping one or more ropes on hand ready to be put on at any time, and of careful inspection of all couplings, the winding gear, sheaves, etc.

At many collieries the management is all that could be desired, but at some workings operated by individuals, and even at some worked by large corporations, the management is extremely lax.

That careful periodic inspection of the pumps, pump rods, cage guides, safety catches, mine and shaft timbering, mine roads, &c., should be made by the superintendent or some other person to act as a check on the mine boss, is generally admitted, and some of the larger operating companies have now perfected such a system of inspection.

At some collieries the condition of the road-beds is very bad. It is seldom possible to construct and maintain a good railroad inside of any mine, but dirty road-beds, defective joints, broken switches, hills and hollows, are never signs of true economy. Clean well-kept road-beds diminish the risk of accident to men, mules, and cars, and reduce the cost of haulage and the wear and tear on rolling stock.

Colliery employés.

The force of employés at large collieries may be divided into several classes with entirely independent duties. This division differs in different districts and at different collieries.

The following list embraces the different classes of laborers and artisans usually recognized:

Under-ground.

Miners.
Miners' laborers or helpers.
Contractors, (miners' laborers, &c.)
Drivers.
Door-boys,—door-tenders, (trapper boys.)
Masons and carpenters.
Track layers and repairsmen, (laborers.)
Laborers.
Stable boss—hostlers.
Fire boss.
Mine boss, inside boss.
Bottom men.
Engineers, (pump-men, inside slope-men, &c.)
Blacksmiths. (?)

At the Surface.

Mine boss, (outside boss.)
Engineers, (winding, pumping, &c., engines.)
Stokers, (firemen.)
Carpenters, (car repairers.)
Laborers.
Drivers.
Head-men,—top-men, (landers.)

Breaker boss.
Platform men, (1st and 2nd.)
Third class men, (laborers.) Breaker.
Slate picker bosses.
Slate pickers, (old men and boys.)
Machinists. (?)
Blacksmith and machinist.
Culm bank-men and drivers.

The blacksmiths, masons, carpenters, machinists, &c., are employed both in the mine and at the surface as circumstances require.

At some collieries there is a foreman who has general management of the colliery, the mine boss being subject to his orders; a more common plan is to make the mine boss the chief of all operations above-ground (as well as inside the mine) with the exception of the breaker, which is managed by the breaker boss. Sometimes the mine boss is also required to exercise a general supervision over the breaker hands.

The fire boss is subordinate to the mine boss; is an assistant to him in fact, but at some collieries the mine boss is also required to perform the duties of the fire boss.

Other collieries are managed by two bosses: an "inside (mine) boss" and an "outside boss," but this arrangement does not always work satisfactorily because it is difficult to find two bosses that will work harmoniously together.

For large collieries the best plan seems to be that first mentioned, viz: To place the colliery under the control of a superintendent or foreman, with the mine boss next in authority beneath him.

When the mine is fiery one or two fire bosses are necessary and they are both in a measure subordinate and assistants to the mine boss.

Discipline.

Insubordination in the force employed underground should be treated with military severity. It can only be viewed in the light of a crime, and should be punished accordingly.

Simple insubordination does not furnish ground for legal action, but the punishment in the hands of colliery managers,—prompt discharge,—cannot be too quickly visited upon willful disregard of any order, however trivial.

It is not competent for the miner or laborer to decide whether ignoring instructions, breaking a prescribed rule, or negligence, will endanger himself or his co-workers. Absolute and immediate obedience, unfailing adherence to all colliery rules, and faithful performance of his duties, are essential to the safety of those employed underground.

Such is the opinion of several of our most prominent mine superintendents, but they find it almost impossible to carry out, even in part, the policy they would endorse in dealing with underground employés.

The same difficulty has been experienced in other mining districts, and can not be removed while artisans and laborers maintain their present attitude towards corporations and individuals supplying them with employment.

Colliery managers and superintendents are not infrequently prevented from discharging insubordinates by the hostile attitude of other employés. The latter commonly regard a discharged miner as a persecuted individual,—a martyr,—and will probably continue to feel in this way until they are educated up to a belief in the common interests of capital and labor,—the mutual dependence of each upon the other.

Whether the passage of a law making an employee liable to criminal prosecution for violation of any rule prescribed for the government of those employed underground, would enable colliery managers to maintain proper discipline is questionable; but that some method (other than that of discharge) of enforcing proper discipline is needed, there can be no doubt.

Colliery rules.

At a large number of collieries a code of rules for the government of employees has been adopted. Several companies have a uniform code of rules, others have regulations framed by the general manager, mining engineer, or superintendent, and at some collieries these rules are printed and posted in conspicuous places.

A code of rules adopted by the operators and superintendents of collieries in the eastern district of Wyoming and Lackawanna, December 24, 1881, is reprinted below from the Mine Inspectors' report.

A shorter set of rules might be more efficient, or a condensed abstract embracing the more important features, for posting in conspicuous places both inside and outside the colliery.

Code of Rules.

"All persons employed in or about this colliery are hereby notified that the following rules and regulations have been adopted for the purpose of preventing injury to persons or property from negligence or carelessness of the employees.

"The attention of each class of workmen is hereby called to the duties assigned them; they are also requested to do all in their power to avoid all unnecessary risk in following their daily avocations.

Mine-boss.

"It shall be the duty of the mine boss to direct and generally supervise the whole working of the mine. He shall instruct the workmen in their several duties and vocations.

"It shall be his special duty to keep the work in proper shape as it advances. He shall keep a careful watch over the ventilating apparatus, air-ways, traveling-ways, pumps and sumps, and shall see that the miners timber their places properly as they advance, and see that they keep their places safe from danger of loose coal, slate, or rock falling upon them. If he shall find a place in a dangerous condition, it shall be his duty to give orders to have it secured by taking down or propping up the loose material, with the least possible delay; or, if necessary, he shall stop the mining of the coal at once, until it is secured. He shall also see that the signaling arrangements from bottom to top and from top to bottom of the shaft or slope are kept in good condition. And he or his assistants shall examine carefully the workings generating explosive gas every morning before the miners enter the mine; and shall ascertain that the mine is free from danger before the workmen are allowed to enter. He or his assistants shall go over the mine every evening and see that the doors along the air passages are properly closed. It shall also be his duty to measure the ventilation at least once a week at the inlet and outlet; also, at or near the face of all gangways, and the measurements to be reported to the inspector once per month.

Fire-boss.

"It shall be the duty of the fire boss to examine carefully every morning every place in the mine where explosive gas is evolved, and see that it is in a fit state for men to work therein before they are allowed to enter the mine. If any of the working places are in an unsafe condition, he shall notify the parties who work therein by danger-signal or otherwise, and they shall be governed by his advice in the absence of the mine boss.

"If explosive gas is found in any of the working places, he shall not allow the men to enter said place or places until he is present to expel or see that it is expelled safely. When a signal board is furnished, he shall mark opposite the number, name, or letter (by which the party is known who works in said place) a mark thus, X, indicating danger. It will then be the duty of all persons working in said place to immediately ascertain the cause of danger, and no one will be allowed to enter such place until authorized by the mine boss or fire boss.

"He shall also mark the date of the month with chalk upon some conspicuous place at the face of each place examined, every morning, and shall be located at some convenient place designated by the mine boss, where he may be seen after his examining tour by every person working in the mine, and there find out the condition of their working place. Any miner or laborer going into his working place where explosive gas is evolved without ascertaining in person the condition of the same, shall be stopped at once, and the same reported to the mine boss. It shall be his special duty to see that all stoppings, doors, brattices and airways are kept in proper condition, and he shall report any defect which he may find in them to the mine boss. He shall also see that all the safety lamps used in the mine are kept in good order ; also, keep a careful watch over the ventilation.

Driver-boss.

"The driver boss shall see that the drivers are at the stables in proper time in the morning, and ready to begin work at the appointed time. He must see that the mules are regularly fed and watered, and properly attended to, and must see that the mules are not driven up steep grades without frequently resting them. He shall see that the mules are not unnecessarily whipped or abused.

"If the safety of persons or animals require a safety-block or latch to be thrown across the track, near the face of working places, he shall see that one or the other be put on at once. He shall not allow door boys to leave their doors except by permission of himself or mine boss.

Duty of miner.

"It shall be the duty of every miner employed in the mine to examine the roof or other overhanging material in his working place as soon as he shall enter the same in the morning, and if found unsafe he shall immediately take down or prop up the loose material, and see that it is in a safe condition for himself and laborers to work therein. No miner shall leave his place in an unsafe condition when his laborers are allowed to work after he has gone home. If the mine boss shall order bad roof to be taken down, or shall order props to be set under the same, it shall be the duty of the miner to attend to the same without unnecessary delay. It shall also be the duty of the miner to take proper care of his powder from the time it leaves the powder house until it reaches his working place in the mine, at which place it must be properly taken care of, and kept in a box, with cover to place over it when the miner is not present. This box must be kept well back from the roads. When the miner is making a cartridge he shall keep his lamp at least four feet away from the nearest part of his box. Said lamp shall be placed upon that side of the box which the current of air would carry a spark from the lamp away from the box. He shall not be allowed to make a cartridge with a lamp upon his head, or his pipe in his mouth, nor shall he set his lamp upon his box. When charging a hole, if the cartridge sticks, he must take it out of the hole carefully, and either make the cartridge smaller or enlarge the hole, so that he may be able to push it easily into it. No ramming of cartridges with a drill will be allowed.

"When driving an entrance between two chambers it shall be the duty of the miner, before firing a shot, to give timely notice to the men in the chamber towards which he is driving, so that they may find a place of safety. They shall also guard the passages on either side of their place, at every shot, so that no person may come unawares upon it when about to fire. They shall also be careful not to go back too soon to a shot which seemed to have missed fire.

"When a shot has been fired, they shall take great care to examine the roof and coal, and see that they are in a safe condition before they go to work under them.

"They shall also see that their car is at safe distance before firing a shot. Before loading their car, they, or their laborers, shall see that no tools, powder, or other material is left in their car. They shall also see that the car is properly blocked and spragged before starting to load it, and after every shot they must see that the road is left clear before the mule shall be allowed to draw the car to the end of the road near the face.

"Where explosive gas is evolved they must learn the condition of their working place before entering the same. Where gas is strong and issuing out in great volumes they shall see that no loose coal or culm is left at the face over night, and that no gas is left burning when they leave their place after the day's work is done. Where blowers of gas are found issuing out of the bottom, no culm or gob shall be left in close proximity to it. They shall also guard against all kinds of accidents, which are liable to occur in a mine, and, as far as practicable, they shall keep their props and gob at least two feet from the road.

Laborer or Helper.

"It shall be the duty of every mine helper or laborer to take proper care in running his car from the face to the gangway. He shall see that it is properly spragged, so that no runaway may occur, whereby persons or property may be injured; and he shall not run his car down to the gangway until called upon. When letting a car down, he must not go before it to hold it back, but shall sprag the wheels sufficiently, and, if necessary, push it when it does not run with the proper number of sprags. It shall also be his duty to fill his road properly for the mule to travel in, and, as far as practicable, he shall keep the sides of his road sufficiently clear of culm or other material, so that a person or mule may pass a car with ease. Where head or stopping blocks are provided for cars to rest against, he shall see that they are properly placed upon the road, as he is going up, so that they may be in a proper condition when the car comes down. He shall also look into his car before loading it, and see that no tools, sprags, or other material, excepting coal, is left in it. He shall see that the car is properly secured before commencing to load it, by putting a sufficient number of sprags in the wheels, and, if necessary, he shall place a prop securely against the lower end of the car. He shall devote his time principally to cleaning, preparing, and loading his coal, but, when necessary, he shall help his miner to set props and do any other work which requires his aid.

Headman and Footman.

"It shall be the duty of the headman and footman, at every shaft where men have to ascend or descend, to be at their proper places when the mine shall be regularly at work, or at such times as the mine boss may designate, and they shall see that not more than ten men are allowed to ascend or descend at the same time on any carriage. If more than ten men shall get on at one time, it shall be their duty to call upon some of them to come off. If the person or persons so called shall refuse to do so, they shall report them to the mine boss, whose duty it shall be to punish them as they deserve. They shall not allow any person to step on the carriage after the signal has been given for the engineer to hoist or lower the carriage, nor leave the carriage until it has rested upon the bottom or top.

"It shall also be the duty of the headman or footman to pay strict attention to the signalling apparatus, and see that they are kept in good condition, so that they may, at all times, communicate intelligently with the engineer, and they must see that all of the signals are properly understood between them.

Driver.

"It shall be the duty of a driver to take proper care of his horse or mule, and see that it is properly fed and watered. He must not whip or abuse it unnecessarily, or allow any other person to do so. He shall drive it carefully, and when ascending steep grade allow it to rest frequently. When he leaves his mule or horse at any time, he must be careful to leave it in a place of safety, where it will be secure from runaway cars or other danger. When drawing cars into a place he must be careful not to drive his mule or horse any further than the track is laid, nor into a pile of coal at or near the face, or to leave the car at a place where he has no room to pass it. If the road is in a bad condition for want of filling, he shall report the same to the mine boss. When drawing cars upon a graded road, he shall be careful to sprag or block the cars sufficiently to prevent them from running upon himself or mule. If head or stopping blocks are used at certain points upon the gangway or main road, he shall see that they are properly placed upon the road when going up with the empty cars, so that they may be in a proper position to stop the cars before they go on to the steeper grade.

"If any person abuses his mule or horse he must report the same to the mine boss, nor will they be allowed to delegate any other person to take out or return their mules to the barn, nor drive their mules to or from the barn faster than a walk.

Company hands.

"All company hands must be at their proper places in the morning to begin work at the proper time, and must not leave until breaker stops, or if working full day they are expected to work the ten hours, and they shall see that all instructions given them by the mine boss or foreman are strictly carried out.

Door boy.

"It shall be the duty of a door boy to be at his post at all times when the mine is regularly at work.

"He must not leave his door at the command of any person except the mine boss or other person to whom he may delegate such authority.

Outside foreman.

"It shall be the duty of outside foreman to direct and generally supervise the outside business over which he is placed. He shall see that all machinery connected with the breaking and preparing of coal be properly fenced off as required by law, and he or his assistants shall see that all persons are free from the machinery before the signal is given for the engineer to start, and see that the boys are kept in their proper places, and not allowed to play about the machinery or cars.

Hoisting engineer.

"It shall be the duty of an engineer to keep a careful watch over the machinery, pumps, steam boilers, &c. He shall see that the boilers are properly supplied with water, and that the steam pressure shall not exceed the limit to which the superintendent of machinery or other officer shall consider them perfectly safe to carry. When a fan is used for ventilation he shall keep it running at such speed as the mine boss or superintendent may direct. He shall not slacken its speed unless directed to do so by said officers. If any repairs are to be made to the fan or other parts of its machinery whereby it is required to be stopped, it shall be his duty to give the mine boss timely notice of the same, so that he may have everything left in proper order. He shall also work his engine slowly and with great care when persons are ascending or descending the shaft. He shall also see that the safety carriage is in good order before letting men down in the morning, and must examine the safety-catches, ropes, and cover, and other parts of the machinery daily, and shall run the safety carriage up and down before allowing men to descend the shaft in the morning. He shall not allow persons to loaf in the engine-room, nor shall he engage in conversation when in the act of lowering or hoisting men or coal. He shall also keep a strict watch over the fireman, and see that he attends to his duties faithfully.

Breaker engineers.

"Breaker engineers shall pay strict attention to all signals from the breaker. He shall not start or move his engine until he is satisfied, either by metal tube or other signal, that all persons are free from the machinery. He shall see that no one is allowed to go around any of the machinery for the purpose of oiling or otherwise, except such persons as are authorized by the foreman.

Slate bosses.

"The slate bosses shall keep the boys in their respective places, and not allow them to go around the machinery or cars. They shall use the greatest caution on all occasions when boys or men are cleaning out rollers or screens, and see that every person is clear of the machinery before giving the signal to the breaker engineer to start his engine.

Penalties.

"For the violation of the above rules and regulations, it shall be the duty of the mine boss or foreman to suspend, discharge, or otherwise punish any person who shall carelessly or willfully neglect to attend to the duties assigned to them.

"When damage to property is carelessly or willfully done, the party so offending shall be subject to pay for the full amount of damage, and may be suspended or discharged for the same offense.

General Rule.

"All persons are hereby forbidden to enter any of the old workings without consent of the mine boss or fire boss, or ride upon the cars on any slope or plane, or send out tools upon a car of coal unless they follow them out and take them off the car before ascending the shaft or slope. Any person who opens a door must see that it is properly closed before leaving it. No person shall be allowed to travel upon a slope or plane while the same is in motion. Persons ascending or descending a shaft will not be allowed to enter upon or leave the carriage while in motion, nor shall they be allowed to step on after the signal has been given to hoist or lower the carriage.

"Any person knowing of the unsafe condition of any place or of damage done to the doors, brattices, or stoppings or obstructions in the air-passages, shall notify the mine boss or fire boss as soon as possible after said damage has been done.

"Any person found guilty of carelessly or wickedly injuring animals or other property shall be held liable for the full amount of damage done to the same.

"All persons must familiarize themselves with the above rules, and any person violating any of said rules will be dealt with as the superintendent may direct.

"I approve the above regulations.

(Signed) PATRICK BLEWITT,"
Inspector of Mines, &c.

A clause prohibiting strangers and others not connected with the works, or employés off duty, from loitering in or around the mine and from trespassing upon the premises, is often included in the regulations.

Fire-board.—At fiery mines the miner must learn the condition of his working place from the fire boss, the mine boss, or "gas-board," before going to work in the morning. At some collieries the "gas-board" is kept on the surface, at others at or near the office of the mine or fire boss underground in a conspicuous place.

The gas-board or fire-board, is a black board on which the fire boss records the condition of each working place and heading.

A copy of the record, shown by the gas-board at Hollenback colliery No. 2 on Sept. 19, 1881, is given below.

Working No.		
11	+	
12	+ +	
86	+	
58	+ + +	
104	+	*in headings.*
21	+	*Brattice.*
41	+ + +	“
46	+ +	“
16	+	“
94	+ + +	“
91	+	“
84	+	“
105	+	“
101	+	“

One cross indicates a trace of gas, and the miner is allowed to proceed with his work, first providing for the removal of the gas (by brattice cloth, etc.); *two crosses* show that there is sufficient gas present to require the use of a safety lamp, and great caution on the part of the miner until the gas has been cleared out, which must be accomplished before the miner is allowed to continue his work at the face; *three crosses* signify that the amount of gas is so great that special precautions must be taken in driving it out to prevent the production of explosive mixtures in adjacent or other working places, and the miner is not allowed to enter his working place unless accompanied by the fire boss or an assistant fire boss to superintend the removal of the gas.

Blacksmiths' shops and Stables underground.

In the list of underground employès given on a preceding page we find blacksmiths included. Some large collieries are provided with a complete blacksmith shop underground. A forge fire and anvil are commonly needed for shoeing the mules at collieries where the mules cannot well be taken to the surface for this simple operation, but whether the maintenance of a complete blacksmith shop underground is advisable is a point upon which colliery managers do not agree.

It is generally considered well to locate the smithy near (but not too close) the stables, so that the stable boss can keep a watch over the fire and thus decrease the risk of accident during the absence of the smiths.

The stables, smithy, and office of the mine and fire boss are commonly located near the foot of the shaft or slope.

Colliery book-keeping.

The advantages derived from having minutely detailed accounts kept for each colliery, showing the exact cost per ton so divided that the cost of mining, development, haulage, winding, preparing, etc., can be readily ascertained, are thoroughly appreciated by many of our anthracite colliery managers and mining engineers; but there are still a large number who concern themselves about the results shown by the monthly or yearly balance sheets only in reference to the amount of *net profit*, without caring to make special inquiry into the relative importance or classification of the items that form the annual expense account.

The total cost of mining and preparing as divided under different headings is usually expressed in cents and fractions of a cent (say to two decimal points), not in *percentages* of the total cost.

The cost per ton may be divided under so many heads that the labor of keeping the accounts is too greatly increased, and much difficulty is experienced in attempting a very minute division. Thus the cost of preparing the coal, of pumping, of winding, and of ventilating all contain a portion of one item (viz: *Steam making*, which includes the cost of fuel, of firemen, of boiler water, wear and tear on boilers, etc.); but what portion of this expense should be added to the cost of preparing, of hoisting, of winding, or of ventilating is a problem that can be only approximately solved even after a most tedious investigation of the relative consumption of steam by these different portions of the mining plant.

There are some items, however, that can be readily kept in separate accounts, thus the amount paid for

Royalty, mining, mine timber, labor underground, labor aboveground, gangway and airway driving, other dead-work, rails, sills, etc., keeping live stock, drivers, door-tenders, engineers and pumpmen, headmen and footmen, repairs, cars, wire rope, general supplies (oil, etc.), general superintendence, clerks, office expenses, preparing coal, improvements, etc., etc.

The following table is here reproduced from the report of the Susquehanna, Mineral R. & Min. Co., Summit Branch, and Lykens Valley Coal companies for 1882:

Cost per Ton in Cents.

	Susquehanna Coal Co.	Mineral R. & M. Co.	Summit Branch RR.*	Lykens Valley Coal Co.
Outside:				
Insurance,	0.41	0.53	0.34	0.70
Legal expenses,	0.08	0.08	0.33	0.18
Live stock,	2.17	1.39	2.53	5.34
Office expenses, rent, etc.,	0.20	0.55	0.39	0.48
Preparing coal,	12.41	14.44	15.95	15.86
Repairs and general expenses,	14.46	15.22	10.52	15.03
Royalties,	3.60	39.80		
Shops and repairs,	0.24	2.99	5.15	0.53
Stable expenses,	4.74	7.11	8.16	15.14
Stationery and printing,	0.15	0.22	0.24	0.32
Stock coal expenses,	0.91	0.13		
Superintendence and clerks,	2.25	2.99	4.82	6.11
Taxes,	1.05	2.70	1.97	1.77
Inside:	42.67	88.15	50.40	51.46
Air and gangways,	4.81	14.50	18.43	13.72
Cars, slope and drift,	2.62	1.86	1.37	4.08
Cross headings and shutes,	2.91	7.63	9.14	9.58
Exhaustion of lands,	8.08		5.14	
Hoisting and pumping,		4.82	14.22	25.08
Mining coal,	48.20	48.71	55.69	56.24
Repairs and general expenses,	41.57	22.12	55.15	76.94
Timber and props,	5.18	9.27	9.33	29.29
Improvement and General:	113.37	108.91	168.48	214.93
Breakers,		0.07		0.21
Cars,		0.38	1.34	3.59
General improvements,		15.49	19.83	24.07
Houses,		1.72	0.56	
Railroad iron, ties, etc.,		2.73	2.72	11.33
Slopes,		4.61		
Total cost:		25.00	24.45	39.20
Average cost per ton at mine,	$1.56	$2.23	$1.96	$3.16
Tons of coal shipped,	911,430	416,235	315,228	168,360

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* There seems to be something radically wrong in the calculation of this column. It foots up $2 43.33, but the actual cost per ton is $1.96,—to make the table foot up correctly 20% should be deducted from all the figures of this column.

The above-mentioned report gives the cost of mining exclusive of royalties, taxes, insurance, and exhaustion of lands in former years, thus:

YEAR.	Susquehanna Coal Co.	Mineral RR. and Mining Co.	Summit Branch RR. Co.	Lykens Valley Coal Co.
1877,	$1 13.68			
1878,	1 04.34			
1879,	96.98			
1880,	1 20.99			
1881,	1 22.00	$1 49.3	$1 91.5	$2 53.1
1882,	1 43.00	1 78.2	1 90.4	3 13

A communication from Mr J. H. Harden showing the cost of mining at the Wilkes-Barre Coal and Iron company's mines *in 1872* is here reprinted from "*Coal*" for March 14, 1883.

Cost per Ton in Cents.

Inside expenses:				
Mining, tons,	960,334	$596,166.70	=	$0.72.491
Driving gangways and airways,	.....	95,067.20	=	9.889
Cross-headings,	.....	37,712.12	=	3.926
Shutes and props,	.....	56,361.95	=	5.869
Haulage,	.....	135,614.52	=	14.122
Construction of railroads,	.....	43,966.73	=	4.578
Repairs to railroads,	.....	58,645.73	=	6.106
Mine cars,	.....	56,399.58	=	5.873
Construction of machinery,	.....	2,425 54	=	.252
Repairs to machinery,	.....	14,167.43	=	1.475
Ventilation,	.....	29,398.11	=	3.061
Total inside expenses, tons,	960,334	$1,225,925.61	=	$1.27.642
Outside expenses:				
Preparing coal, tons,	960,334	$271,859.23	=	$0.28.308
Haulage,	.....	45,643.01	=	4.753
Repairs to railroads,	....	11,861.15	=	1.235
Construction of railroads,	....	2,033.50	=	.211
Construction of machinery,	.....	5,620.90	=	.585
Repairs to breakers and machinery,	.....	13,134.34	=	1.367
Repairs to mine-houses,	.....	1,120.56	=	.116
Ventilation,	.....	6,675.96	=	.695
Total expenses outside, tons,	960,334	$357,948.65		$0.37.270
Total expenses inside,	.....	1,225,925.61	=	1.27.642
Royalty on coal mined under lease,	....	198,193.50	=	20.638
Royalty on coal mined from company's lands.	.....	62,572.34	=	6.511
Improvements inside,	.....	321,115.13	=	33.430
Improvements outside,	....	241,256.17	=	25.130
Total tons, cost,	960,334	$2,407,011.40	=	$2.50.621

It will be observed that the cost of ventilation is found under two heads, but the sum of these two items (3.061 and .695=3.756) cannot be assumed as the true cost of ventilation. If we introduce this as a separate sub-division in any statement it is evident that the cost of driving and maintaining all air-ways, headings, and other passages used only for ventilation, the cost of brattice, brattice cloth, stoppings and doors, the wages of door-tenders, &c., must be differentiated from the accounts and added to the cost of driving the fan, or operating a furnace, before we can arrive at the increase in the cost per ton due to ventilation.

The cost of the labor required to make a thoroughly reliable detailed statement of the cost of ventilation, haulage, winding is generally considered to be greater than the advantages derived from such statements, and we consequently find the books of many coal mining companies kept in such a manner that it is almost impossible to obtain the actual cost of any particular branch of mining.

The following table, which has been compiled from the reports of the Bureau of Industrial Statistics, is given for comparison. The collieries included in this table were taken at random from those having a production of more than 75,000 tons per annum.

The figures are only approximate, and in many cases are evidently untrustworthy. It is exceedingly difficult if not impossible to obtain reliable data from many of the individuals and corporations mining coal. Their reasons for withholding exact statements are often of a private nature, and it seems beyond the province of this report to examine into, and disclose to the public the private business of individuals or details of the business of corporate bodies.

*Cost of Mining Coal in 1877.**

NAME OF COLLIERY OR PERSONS OPERATING SAME.	Product in tons.	Amount paid for wages.	Amount paid for materials.	Cost per ton.
Thomas Coal Co.,	89,852	$81,579	$25,662	$1.19
Lawrence, Merkel & Co.,	79,823	97,409	30,650	1.60
Middle Lehigh Coal Co.,	120,141	124,342	24,476	1.24
Gilberton Coal Co.,	82,691	74,473	25,317	1.21
R. Hecksher & Co.,	85,077	80,823	11,761	1.09
Wm. Penn Coal Co.,	150,000	150,000	30,000	1.20
Philadelphia & Reading Co.				
Merriam,	105,759	107,042	31,767	1.31
Ellengowan,	78,089	66,417	24,238	1.16
Plank Ridge,	90,657	81,988	15,306	1.07
West Brookside,	154,544	128,444	60,872	1.22
Indian Ridge,	96,092	97,033	12,414	1.14
Knickerbocker,	99,662	78,985	32,361	1.12
Totals and average,	1,232,387	$1,168,535	$324,824	$1.21

* It will be seen that the amount paid for royalty, insurance, etc., is not included in these tables. I have included only those collieries producing 75,000 tons or more.

Cost of Mining Coal in 1878.

NAME OF COLLIERY OR PERSONS OPERATING SAME.	Product in tons.	Amount paid for wages.	Amount paid for materials.	Cost per ton.
Williamstown Colliery,	268,531	$245,083	$77,443	$1.20
Del. & Hudson Co.'s mines,*	1,652,221	1,386,874	219,550	.97
Hillside I. & C. Co.,	269,496	261,500	49,800	1.15
Roaring Brook,	77,305	73,151	18,807	1.19
Susquehanna Coal Co.,	408,452	374,840	103,296	1.17
Mineral R. & M. Co.,	102,400	80,000	41,000	1.18
Cameron,	154,294	125,000	69,000	1.26
Kohinoor,	112,000	95,200	22,400	1.05
Wm. Penn,	123,000	150,000	50,000	1.62
Kehley Run,	90,070	89,670	18,649	1.20
Kalmia,	94,912	80,080	22,400	1.08
Philadelphia & Reading Co.				
Hammond,*	101,966	76,597	23,211	.98
Indian Ridge,	122,368	106,690	22,412	1.05
Knickerbocker,*	104,882	75,420	12,806	.84
Mt. Carmel,	122,455	122,240	20,242	1.16
West Brookside,*	282,264	192,099	39,929	.82
Totals and average,	4,086,616	$3,434,444	$810,945	$1.04

* These figures greatly reduce the average shown by the footings.

Cost of Mining per Ton in 1879.

NAME OF COLLIERY.	Product in tons.	Amount paid for wages.	Amount paid for materials.	Cost per ton.
Williamstown,	259,889	$278,607	$70,993	$1.34
Spring Mountain,	175,000	175,000	50,000	1.29
Cameron,	163,000	118,000	44,000	.99
Big Mountain,	147,955	125,625	36,649	1.09
Kohinoor,	165,000	140,000	24,000	.99
Wm. Penn,	125,000	125,000	50,000	1.40
Lawrence,	102,739	96,429	26,345	1.19
Philadelphia & Reading Co.				
Bast,	139,431	123,073	31,560	1.11
Conner,	132,056	109,632	27,940	1.04
Knickerbocker,	173,715	130,709	36,951	.91
Mt. Carmel,	182,922	168,906	27,074	1.07
Thomaston,	123,849	122,583	36,216	1.28
Totals and average,	1,890,556	$1,713,564	$461,728	$1.15

The following table shows the average cost per ton of mining from 1873 to 1882 inclusive, at the collieries of the Philadelphia and Reading Coal and Iron company:

Average Cost Per Ton.

YEAR.	Tonnage P. & R. C. and I. Co.	Cost.
1873,	1,348,838	$2 51
1874,	1,374,796	2 44.8
1875,	1,510,572	1 86.7
1876,	1,853,364	1 35.4
1877,	3,794,528	1 03.9
1878,	2,727,608	1 23.7
1879,	4,269,929	1 14.3
1880,	3,460,464	1 43.6
1881,	3,937,607	1 49.7
1882,	4,111,830	1 47.3

Table showing the Tonnage, Expenses, and Average Cost per ton of Mining Coal at collieries worked by the Philadelphia and Reading Coal and Iron Company, from 1873 to 1882, inclusive.

MONTHS.	FOR THE YEAR 1873.			FOR YEAR 1874.			FOR YEAR 1875.		
	Tonnage.	Expenses.	Cost per ton.	Tonnage.	Expenses.	Cost per ton.	Tonnage.	Expenses.	Cost per ton.
December, . . .	92,524 03	$249,352 40	$2 71.3	80,212 05	$217,527 71	$2 71.2	82,893 15	$155,419 39	$1 87.4
January, ...	11,353 11	82,202 45	7 31	10,915 01	91,066 26	8 34.3	5,907 14	41,840 96	7 08.2
February, . . .	29,388 03	151,482 00	5 15.4	50,816 07	176,780 49	3 47.9	5,411,09	50,347 84	9 30.4
March, ...	92,698 01	257,187 54	2 77.4	110,748 12	312,285 86	2 81.9	6,413 17	67,937 75	10 59.2
April, ...	128,505 18	325,924 74	2 53.6	129,323 10	309,952 99	2 39.6	5,790 09	72,597 08	12 53.8
May, ...	123,492 10	316,948 91	2 57	142,777 00	321,971 28	2 25	9,256 09	83,596 61	9 03.1
June, ...	127,206 09	318,772 52	2 50.5	145,759 15	319,998 19	2 12.6	92,974 13	215,645 03	2 31.9
July, ...	126,793 12	312,720 94	2 46.6	64,415 19	211,997 15	3 29	260,312 04	424,431 33	1 63
August, ...	140,481 18	328,826 46	2 34	88,309 17	236,610 80	2 67.9	256,546 12	441,424 95	1 72
September, . . .	160,957 06	349,221 18	2 17.6	158,228 05	352,044 78	2 21.9	266,150 19	452,129 57	1 59.7
October, ...	167,249 03	362,041 23	2 16.3	226,267 16	450,200 70	1 98.9	290,019 06	447,989 68	1 54.4
November, . . .	148,196 14	530,469 31	2 23	167,016 06	364,472 16	2 18.2	228,895 04	368,249 32	1 60.8
Total, ...	1,348,838 08	$3,385,149 68	$2 51	1,374,790 16	$3,364,908 37	$2 44.8	1,510,572 11	$2,821,609 51	$1 86.7

Table showing the Tonnage, Expenses, &c.—Continued.

MONTHS.	FOR ELEVEN MONTHS ENDING NOVEMBER 30, 1876*.			FOR TWELVE MONTHS ENDING NOVEMBER 30, 1877.*			FOR TWELVE MONTHS ENDING NOVEMBER 30, 1878.*		
	Tonnage.	Expenses.	Cost per ton.	Tonnage.	Expenses.	Cost per ton.	Tonnage.	Expenses.	Cost per ton.
December, . . .	. . . .	. . . .	. . . .	198,411 14	$219,449 94	$1 10.6	361,829 06	$344,038 73	$0 95.1
January, ...	42,274 19	$132,015 09	$3 12.2	146,082 05	198,733 45	1 36	96,935 03	230,859 77	2 38.1
February, . . .	26,882 10	133,765 29	4 97.6	133,114 00	229,705 34	1 72.6	65,680 18	205,538 31	3 12.9
March, ...	49,097 08	183,750 29	3 74.4	205,713 04	310,245 07	1 50.8	89,324 06	193,271 47	2 16.4
April, ...	208,686 07	309,703 93	1 48.4	324,382 06	353,458 40	1 08.9	189,983 03	240,832 14	1 26.7
May, ...	206,103 00	276,358 30	1 34	380,714 12	386,974 63	1 01.7	240,057 06	273,676 01	1 14
June, ...	117,746 16	173,274 47	1 47.1	375,710 10	349,610 54	93	333,193 06	358,308 69	1 07.5
July, ...	122,469 01	169,276 49	1 38.2	376,249 12	335,370 90	89.1	191,880 03	262,363 97	1 36.7
August, ...	177,767 02	229,404 88	1 29	420,398 09	418,497 56	99.5	341,129 03	377,853 02	1 10.8
September, . . .	264,637 19	270,664 27	1 02.3	419,602 06	404,810 39	96.5	139,736 11	208,911 29	1 49.5
October, ...	316,974 14	316,696 11	99.9	441,902 05	419,780 08	95	299,268 02	330,792 24	1 10.5
November, . . .	320,724 05	314,574 12	98	372,247 13	315,925 41	84.9	378,590 14	347,558 08	91.8
Total, ...	1,853,364 01	$2,509,483 24	$1 35.4	3,794,528 16	$3,942,561 71	$1 03.9	2,727,608 01	$3,374,003 72	$1 23.7

Table showing the Tonnage, Expenses, &c.—Continued.

Months.	For year ending Nov. 30, 1879.*			For year ending Nov. 30, 1880.*		
	Tonnage.	Expenses.	Cost per ton.	Tonnage.	Expenses.	Cost per ton.
December,	214,046	$266,119 76	$1 24.3	316,044	$423,957 00	$1 34.1
January,	208,188	245,619 49	1 18	236,375	337,811 00	1 59.8
February,	260,001	286,632 50	1 10.2	169,502	306,214 00	1 80.6
March,	323,378	349,069 60	1 07.9	254,792	396,204 00	1 55.5
April,	373,040	414,626 17	1 11.1	318,726	517,265 00	1 62.3
May,	418,642	454,113 45	1 08.5	247,777	405,186 00	1 63.5
June,	398,195	472,319 68	1 18.6	266,599	373,597 00	1 40.1
July,	412,490	468,364 80	1 13.6	211,634	332,836 00	1 57.3
August,	413,270	479,592 10	1 16	244,335	334,312 00	1 40.9
September,	419,241	493,109 09	1 17.6	439,615	545,679 00	1 24.1
October,	440,464	498,724 13	1 13.2	361,198	475,446 00	1 31.6
November,	388,969	453,077 14	1 16.5	393,861	472,863 00	1 20
Total,	4,269,929	$4,881,368 00	$1 14.3	3,460,464	$4,971,371 00	$1 43.6

Table showing the Tonnage, Expenses, &c.—Continued.

Months.	For year ending Nov. 30, 1881.*			For year ending Nov. 30, 1882.*		
	Tonnage.	Expenses.	Cost per ton.	Tonnage.	Expenses.	Cost per ton.
December,	209,223	$353,138 00	$1 68.3	383,939	$597,409 00	$1 55.6
January,	222,234	363,494 00	1 63.6	240,432	393,671 00	1 63.7
February,	235,161	382,777 00	1 62.8	202,611	337,334 00	1 66.5
March,	283,874	462,470 00	1 62.9	270,238	437,786 00	1 62.0
April,	253,346	416,425 00	1 64.4	298,187	474,117 00	1 59
May,	298,377	459,762 00	1 54.1	285,950	457,522 00	1 60
June,	350,403	519,745 00	1 48.3	339,179	488,418 00	1 44
July,	392,375	552,022 00	1 40.7	396,101	560,483 00	1 41.5
August,	434,643	593,927 00	1 36.7	416,405	571,661 00	1 37.3
September,	416,459	601,790 00	1 44.5	377,175	539,362 00	143
October,	420,115	609,145 00	1 45	441,813	593,798 00	1 34.4
November,	421,391	581,859 00	1 38.1	459,795	606,929 00	1 32
Total,	3,939,607	$5,895,555 00	$1 49.7	4,111,830	$6,058,490 00	$1 47.3

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* Note the relation of the monthly variation in the tonnage to the cost per ton.—H. M. C

CHAPTER XXI.

Mine Surveying and Mapping.

The law requires all anthracite colliery owners to prepare maps of all workings on a scale of one hundred feet to an inch for the use of the mine inspector.

This scale is rather too large for convenient use, and we consequently find most of the working maps used for reference constructed on a scale of two or three hundred feet to an inch.

These maps generally show all the most important surface features, the location of all buildings, streams, roads, railroads, &c., as well as the under-ground workings. The latter are commonly drawn in blue, red, or green ink. When several beds are worked, the workings, and all data relating to the workings on each bed are shown by a separate color.

This plan is especially necessary when the workings on one seam over or underlie those opened on another bed.

In addition to the general map showing all the workings, separate maps showing the workings on each seam are usually made.

The survey lines are plotted with a vernier protractor, or a protractor of very large size, and the location checked by latitude and departure calculations when a high degree of accuracy is desired.

Tracings or blue prints of the workings are from time to time furnished the mine boss.

When not in use, the mine maps and other important drawings are stored in large fire-proof vaults.

Some engineers use an ordinary pocket note book to record the survey notes, but as these books soon become filthy, and the writing badly blurred, the survey notes are copied into a series of larger volumes kept in the draughting room ;

others use tablets of note paper, tearing off each sheet as soon as it becomes filled with notes, or is badly soiled, and filing it carefully in a pocket-book, or other receptacle. The notes are copied from these loose sheets and transcribed in the office record books, and the sheets filed away for future reference.

With the exception of the work done by the United States Coast Survey, there is probably no other series of surveys in America that will compare in accuracy with the anthracite mine surveys*.

While we cannot expect, nor could we desire, more satisfactory survey work than that now done at nearly all anthracite mines, in one respect many such surveys are gravely defective. I refer to the absence of a carefully connected system of tide water levels.

Many mine maps contain no levels, and the relative elevations of different parts of the workings can only be roughly obtained from the dip of the coal as marked upon the map, from estimates of the gangway grades, and by measuring the depth of shafts, etc.

What is needed is a carefully connected system of elevations based upon a common datum,—preferably that of mean tide.

Several companies having all of their mines leveled, and the elevations marked upon the mine maps, have not a connected system,—the elevations at each mine being based upon a separate datum.

A few of the larger operating companies have had all of their mine-levels connected, and reduced to a common datum plane.

Accurate levels are as essential as accurate maps, and the law requiring the latter seems especially defective in making no provision for the former.

It seems entirely unnecessary to dilate upon the necessity of accurate mine levels. Their value is thoroughly appreciated by many colliery owners, and by nearly all superintendents and engineers, and we can confidently look forward to a time in the near future when all working mines will be as carefully and thoroughly leveled as they are now surveyed and mapped.

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*Some tunnel surveys, etc., may rival them in accuracy, but these are only special pieces of engineering work.

The American transit, with vertical limb, with vernier for taking vertical angles, mounted on a tripod with adjustable (or with two sets of) legs, is used to the almost entire exclusion of other instruments.

Rods of various forms are used ; the Philadelphia rod is preferred by many for outside work, and a somewhat similar, (but much smaller adjustable rod for inside work, the target being perforated by a slit at the center opposite the lamp flame ; when levels are not taken, the plumb line and lamp are often used without a rod.

Measurements are now made by some engineers with a steel tape or wire. Some engineers prefer using very long wires or tapes,—500 feet long,—thus eliminating a large part of the error common to all chain measurements, but the chain is still preferred by many to any other method of measuring.

For inside work the transit horizontal limb is preferably graduated throughout the whole circumference from 0° to 360°, but a second series of figures is often added to show the ordinary N., E., S. and W. points, and the corresponding quadrant graduations. A non-adjustable six-foot rod is often used.

In addition to these instruments, two plummet lamps, a plumb-bob and cord, and a transit point are used, as well as ordinary mine lamps (made of copper) to furnish sufficient light.

When the levelling is carried on independently, a large well adjusted Y level, and the target rod with slit above mentioned, form the levelling equipment.

Chain measurements, except for unimportant branch lines, breast measurements, etc., have been abandoned by some engineers for the steel tape method introduced by Mr. Eckley B. Coxe. A description of this method, contained in a paper read by Mr. Coxe before the Institute of Mining Engineers (vol. II, page 219) is here reprinted :

Method of measuring in Mine Surveys.

"In making surveys in the anthracite coal regions of Pennsylvania, the ordinary engineer's chain (50 or 100 feet long) is generally used, both above and below ground. Sometimes, where it is difficult to chain, as, for instance, across a chasm, a wire is stretched from one station to the other, the distance is marked on the wire and its length is then measured with the ordinary chain. Having had occasion lately to make some surveys where it was necessary to determine with great accuracy the position of the land or property line, not only in the gangways or levels, but also in the breasts or chambers, the coal on the north side of the line belonging to one party and that on the south side to another, and as it is very difficult to measure up the breasts or slopes with accuracy, and to make the proper allowance for the pitch of the vein (the true horizontal distance being, of course, the product of the distance measured with the chain by the cosine of the angle of inclination of the chain,) and as the ordinary method of chaining up or down steep slopes on the surface, by holding a portion of the chain horizontal and plumbing down from the high end, would in most cases be very difficult and dangerous, and sometimes impracticable, I determined to adopt a new plan which would do away with most of the above difficulties, and by which I could eliminate many causes of error from my ordinary chaining.

"My first idea was to have a fine steel-wire rope, about 300 feet long, stretched as much as possible in making, so as to do away as well as I could with that source of error, and then to have it graduated every ten feet. I proposed using small brass tags of different shapes to designate the different hundred feet, thus:

0—100 a triangle.
100—200 a square.
200—300 a circle, etc.

"The numbers of the ten feet spaces were to be marked by drilling small holes in the tags. I intended to use this for the principal lines of my surveys and to use the chain only for lines which were not of great importance.

"When I called upon Mr. Heller (of Heller & Brightly, the instrument makers, of Philadelphia) to order this measure, he suggested that it would be better to use instead of a wire rope, which would stretch, the bands which are manufactured for hoop skirts ; they are made of tempered steel, are very light, and will not stretch sensibly. After consultation with him, I decided to have the tape measure constructed which is now before you. It is 500 feet long and weighs 2 ℔. 7½ oz. It is a ribbon of tempered steel, 0.08 inch wide, 0.015 inch thick. At each 10 feet a small piece of brass wire is soldered across the tape, the solder, which is white, extending about one inch on each side of the wire. In the latter a small notch is filed, which marks the exact point where the ten feet ends. The exact distances from the zero point of the tape are marked upon the solder by countersunk figures. The white solder enables one to find the ten feet notches very easily, and, no matter how dirty the tape may be, by wiping off the solder with the finger, the distances are easily read, as the countersunk figures, being filled with dirt, stand out upon the white ground of the solder. The 0 and 500 feet marks are not at the end of the tape, but near it, and are also denoted by a notch filed in a wire soldered to the tape.

"The tape is wound upon a simple wooden reel, ten inches in diameter, which is held in one hand and turned by the other. At first some difficulty is experienced in winding up the tape, but a little practice soon overcomes it. Two brass handles, which can be detached, accompany the tape and are carried upon the reel.

"*Description of a Survey made with the Tape.*—The instruments used were one of Heller & Brightly's new 11-inch transits, two plummet lamps, the 500 feet tape and a 5-foot wooden rod divided into feet and tenths. The latter is used to measure the distance from the nearest ten feet to the station. There were two closed sets of lines or surveys, one set entirely above ground, but through the swamps and brush of the anthracite coal region, and one partly above ground and partly in the mines. The latter began at a point in the swamp, went overground 2400.57 feet to the mouth of the slope, then down the slope (pitch 37°), 276.99 feet (horizontal distance), then along the gangway 4272.01 feet, which formed one half of an ellipse, then up through a breast (pitch about 34°) 275.44 feet (horizontal distance) to the bottom of an air-shaft, then by two plumb lines to the surface, and then through the swamp 141.83 feet on the surface to the point of beginning. The length of the periphery of the first closed figure was 6660.19 feet; that of the second 7366.84 feet. Tables I and II show the details and calculations of the two surveys:

TABLE I.

Station.	ANGLE.		Reduced Angle.	Distances.	Positive Sine.	Negative Sine.	Positive Sum of Sines.	Negative Sum of Sines.	Positive Cosine.	Negative Cosine.	Positive Sum of Cosines.	Negative Sum of Cosines.
	Right.	Left.										
1	0°04′	. . . .	0°04′	664.97	C.78	. . . .	0.78	. . .	664.75	. . . .	664.75	. . .
2	0°47′		0°51′	711.55	10.56	. .	11.34	. . .	711 47	. . . .	1376.22	. . .
3		0°52′	− 0°01′	408.60		0.12	11.22	. . .	408 16	. . . .	1784.38	. .
4	0°33′		+ 0°32′	567.25	5.28		16.50	. .	567.22		2351.60	. . .
5	. .	179°32′	−179°00′	186.05		3.24	13 26	. . .	. . .	186 02	2165 58	. . .
B	. . .	31°12′	−210°12′	88 42	44 48	. .	57.74	. . .	. . . .	76.42	2089.16	. .
B1	. . .	19°39′	−229°51′	389.50	297.72	. . . .	355.46	. . .	. . .	251.15	1838.01	. . .
B2		9°36′	−239°27′	631.00	543.41	. . . .	898.87	. . .	. . . .	320.73	1517 28	. . .
B3	4°06′	. .	−235°21′	381.25	313.62	. . .	1212.49	. . .	. . . .	216.77	1300.51	. . .
B4	35°55′	. .	−199°26′	752 50	250.37		1462.86	. . .	. . . .	709.63	590.88	
B5	62°39′		−136°47′	294.80	: .	201.87	1260.99	. .	. . .	214.84	376.04	
5	.	9°01′	−145°48′	527.20	. . .	296.33	964.66	.	.	436.04		60.00
4	86°27′		− 59°21′	464.85	. . . .	399.91	564.75	. . .	236.97		176.97	.
3	. . .	44°17′	−103°38′	210.05	. .	204.14	360.61	. .	.	49.50	127.47	. . .
2	. . . .	5°51′	−109°29′	382.20	. . .	360.32	+0.29	. .	. .	127.47	0.00	. . .
					1466.22	1465.93			2588.57	2588.57		
				Diff'rnce+0 29				Diff'ce 0.00				

TABLE II.

B	. . . .	30°12′	−210°12′	88.42	44.48	. .	57.74	. .	. . . .	76.42	2089.16	. . .
B1	. . .	19°39′	−229°51′	389.50	297.72	. .	355.46	. . .	. . . .	251.15	1838.01	. . .
B2		9°36′	−239°27′	631.00	543.41	. . .	898.87	. . .	. . . .	320.73	1517.28	. . .
B3	4°06′	. . . .	−235°21′	381.25	313.62	. . .	1212.49	. . .	. . .	216.77	1300 51	. . .
B4	35°55′	. . .	−199°26′	752.50	250 37		1462.86	. . .	. . . .	709 63	590.88	. . .
B5	31°20′	. . .	−168°06′	157.90	.	32.56	1430.30	. . .	. . . .	154.51	436.37	. . .
O	73°10′	. . .	− 94°57′	276.99	. . . .	275.96	1154.34	. .		23.90	412.47	. .
1	88°44′		− 6°13′	651 98	. . .	70 60	1083.74	. .	648 29	. .	1060.76	. . .
2		11°35′	− 17°48′	157 06	. . .	48.01	1035.73	. . .	149.54	. . . .	1210.30	. . .
3	6°41′		− 11°07′	281.50	. . . .	54.28	981.45	. .	276 22	. . . .	1486.52	. . .
4	8°21′		− 2°46′	86 90	. . . .	4.19	977 26	. . .	86 79	. . . .	1573.31	. . .
5	.	9°26′	− 12°12′	192 91	. . . .	40.77	936.49	. .	188.55	. . . .	1761.86	. . .
6		4°31′	− 16°43′	208.12	. . . .	59.86	876.63	. . .	198.86	. . . .	1960.72	. . .
7	2°17′		− 14°26′	235 85	. . . .	58.79	817.84	. . .	228.40	. . . .	2189.12	. . .
8		7°35′	− 21°59′	476.32	. . . .	178.30	639 54	. . .	441 68	. . . .	2630.80	. . .
9	4°09′		− 17°52′	101.20	. . . .	31.05	608.49	. .	96.32	. . . .	2727.12	. . .
10		7°56′	− 25°48′	235.64	. . . .	102.56	505 93	. . .	212.15	. . . .	2939.27	. . .
11	18°50′		− 6°58′	370.85	. . .	44.98	460.95	. .	368.11	. . . .	3307.38	. .
12	. .	51°46′	− 58°44′	86.40	. . .	73.85	387.10	. . .	44.84		3352 22	. . .
13	. . . .	44°30′	−103°14′	99.95	. . . .	97 30	289.80	. . .	. . . .	22.88	3329.34	. . .

14	.	78°51′	−182°05′	316.50	11.51		301.31	. .	. . . . .	316.30	3013.04	.
15	14°59′	. .	−167°06′	151.38	. .	33.80	267.51	. .	. . . . .	147.56	2865.48	. . .
16	13°08′	.	−153°38′	107.95	. . .	47 38	220.13	. . . .	. . . . .	97.00	2768 48	. .
17	.	15°30′	−169°28′	123.36		22.55	197.58	. . . .	. . . . .	121.29	2647.19	. .
18	. . . .	13°15′	−182°43′	209.60	9.93		207.51	. .	. . . . .	209.37	2437.82	. .
19		4°40′	−187°23′	178.54	23.00		230.51	. .	. . . . .	177.65	2260.17	. . .
20G	58°10′	. . . .	−129°13′	21 91	.	16.98	213.53	.	. . . . .	13.85	2246 32	. . .
F	24°44′	. .	−104°29′	173.71	. .	167.56	45.97			43.44	2202.88	. .
E	34°10′	. . .	− 70°19′	79.82	. . .	73.79		27.82	26.39	.	2229.27	. . .
D	21°59′	.	− 48°20′	9.08	. . .	6 78		34.60	6 04	.	2235.31	. . .
C	.	19°08′	− 67°28′	25.95		23.97		58 57	9.94		2245.25	. . .
A	.	154°46′	−222°14′	106.80	71.81	.	13.24		. . .	79.05	2166.20	. . .
B	43°15′	. . .	. . . . . .	. . . . .		. . .	.	. .	. . .	.	. . . .	. . .
					1565.85	1565 87			2982.12	2981 50		
					Diff'rnce−0.02			Diff'ce+0.62				

"From these we see that the total errors were in the

	Sine.	Cosine.
First case, ...	+0.29	0.00
Second case, ...	—0.02	+0.62

"This is very accurate work for this kind of mine surveying. We made three other surveys on the same property with equally good results.

"In measuring with the tape it is better to have at least three men, one at each end and one to take off the distances and note them. The hind chainman should be a reliable man, as he must hold the zero point of the tape exactly at the nail in the stake, or alongside of the cord to which the plummet-lamp is suspended. The front chainman has merely to stretch the tape and to see that it passes exactly over the front station. The third man, who carries the five-foot rod, starts from the rear station and notes the distances of the breasts, &c., as he goes along until he arrives at the forward end, where he notes the distance of the station from the last one. In measuring distances of over 500 feet, a temporary station is made at 500 feet exactly in the line to be measured.

"*Advantages of the Tape.*—First, greater facility in measuring up or down slopes, breasts, &c. Second, greater accuracy in measuring from one station to another, as the tape forms a straight line from one station to another, and as there is no error from the use of pins. Third, the tape does not stretch appreciably.

"*Disadvantages.*—First. It is liable to break unless carefully handled. Second. It is necessary to roll it up and unroll it, when the distances between stations vary much.

"The tape can be easily mended by any watchmaker when it breaks, and Messrs. Heller & Brightly make a small sleeve of brass, tinned inside, in which the ends of the tape, when broken, are slipped and then soldered fast by merely heating the sleeve with a red-hot poker. They also have little brass clamps to fasten on the tape to mark any point which is to be used several times.

"When the men become accustomed to the tape they wind it up and unwind it very quickly.

"There are three sources of error which may be referred to, viz:

I. The extension of the tape by stretching.

II. The shortening of the tape in consequence of the tape assuming the form of the catenary curve.

III. The contraction or expansion due to the change of temperature.

"As stated above, the tape does not stretch appreciably, but this error being in the opposite direction is, to a certain extent, compensated for by the shortening due to the formation of the catenary curve by the tape. I subjoin a table, calculated by my assistant, Mr. Edgar Kudlich, showing the shortening of the tape due to the latter cause. The tension in practice is from 30 to 40 pounds.

TABLE III.

Length of Tape Measure.	True distance when tape is subjected to a tension of (or cord of the catenary curve formed the tape,)					
	10 ℔.	20 ℔.	30 ℔.	40 ℔.	50 ℔.	60 ℔.
100 feet, . . .	99.9894	99.9974	99.9988	99.9993	99.9995	99.9997
200 " . . .	199.9153	199.9791	199.9907	199.9948	199.9967	199.9977
300 " . . .	299.7143	299.9294	299.9687	299.9824	299.9887	299.9922
400 " . . .	399.3268	399.8327	399.9260	399.9583	399.9733	396.9815
500 " . . .	498.6775	499.6732	499.8551	499.9185	499.9479	499.9638

"According to the table given by Haswell for the expansion of steel, a tape measure 500 feet long at 32° Fahr., would become 500.6 feet long at 212°, so that a variation of 60° in temperature would only cause a variation of two tenths of a foot in a 500 feet tape.

"In conclusion, I would advise the use of the tape for all important work, while the chain should be used for filling in details, and where accuracy is not absolutely necessary.

"*Discussion.*—Mr. Coxe remarked in answer to questions that no correction was applied for temperature, and no allowance for stretching of the wire ribbon. He thought its extension was practically *nil.*

"Mr. Raymond commented on the fact that, while mining and surveying instruments of all kinds had been improved so much in recent years as regards accuracy and precision, the method of measuring distances—the chain—had remained the same. Nothing could be inherently more objectionable as a standard of measurement than a chain, composed of links which are liable to wear by friction."

The transit point is a metallic cone used for transferring the location of a station marked in the roof to the floor so that the transit may be set up directly under the spud, or for marking a transit station temporarily on the floor until it can be marked upon the roof.

Nearly all stations are marked in the roof, by driving a nail (often of copper) with a head resembling that of a horse shoe nail with a hole or cross to mark the center, or with a pyramidal head, into the mine timber directly over the station; or by driving it into a wooden plug inserted in a hole drilled in the roof for that purpose. This nail is called a "spud" or "spad."

For work in fiery mines a plummet-lamp surmounted by a glass cylinder and wire gauze arranged after the style of the Mueseler safety-lamp is sometimes used.

While greater accuracy is undoubtedly attained and the risk of serious error diminished by the use of plummet-lamps, a very good degree of accuracy is obtained by the use of an ordinary plumb-bob and line and a hand-lamp held so that the flame will be exactly in front of or behind the cord. In the same way the ordinary Mueseler or other safety-lamp is used with good results in place of the plummet safety-lamp.

The fact that all working mines are provided with two outlets, generally enables the mining engineer to complete the circuit and close the survey of all main lines, and no survey is considered completed nor are its results accepted as established until such closure has been effected.

The main lines of most surveys follow the gangways, and the airways are commonly located by offsets through the crossheadings.

Chapter XXII.

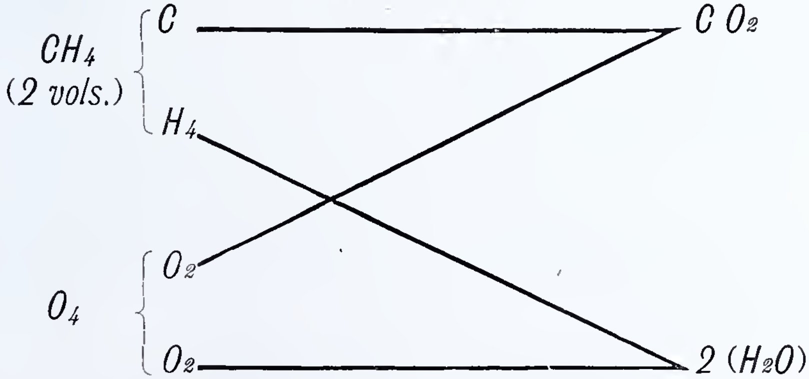
Mine gases and Explosions.

In the preparation of the following material relating to the chemical composition and properties of mine gases, I have made free use of the information contained in works and essays (mostly English) written by acknowledged authorities on the subject.

It is to be regretted that we have no analyses showing the composition of the gases given off by anthracite coals in this country, and until such are obtained, we are necessarily compelled to assume that they are similar to those occurring in other coal-fields.

Fire-damp, often improperly called "*sulphur*" by the miner, is a name applied to any combustible gas found in a mine other than "*white-damp*" or carbonic oxide. In a vast majority of cases it probably consists of marsh-gas (CH_4), but it may also contain hydride of ethyl ($C_2 H_6$), hydride of propyl ($C_3 H_8$) or quartane ($C_4 H_{10}$).*

Light carburetted hydrogen, hydride of methyl, or fire-damp (CH_4) is composed of one atom of carbon and four atoms of hydrogen, requiring therefore for its complete combustion four volumes of oxygen, thus:



(379 AC.)

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* See Thomas on "Mine Gases."

resulting in the production of one molecule of carbonic acid gas and two molecules of water.

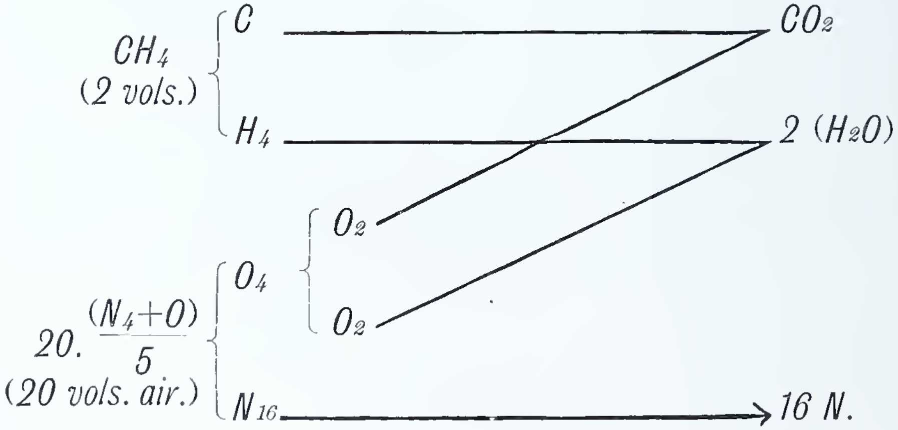
Its specific gravity is 0.5576, and it is eight times as dense as hydrogen. One thousand cubic feet of it weigh at 32° F. and 30 inches bar., 44.665 lbs.

As this gas (CH_4) is not affected by acids nor by any known substance (except chlorine) at normal temperature and pressure, it does not seem probable that it will be possible to render it harmless by chemical combination with another substance.

It is soluble in water to the extent of 3.9 volumes per 100 at 15° C (59° F.)

It is oxidised (slowly burnt) like hydrogen by spongy platinum, (and palladium,) and this property has been utilized in Koener's "Apparat Zur Verzehrung 'Schlagender Wetter'" which will be described further on.

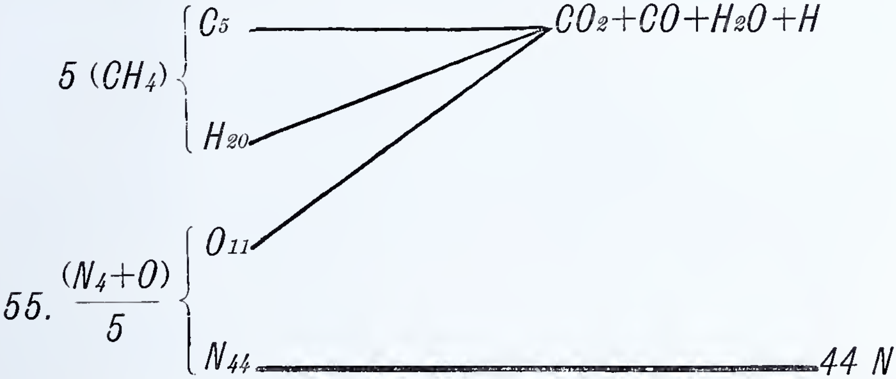
Atmospheric air being considered to consist by volume of four parts of nitrogen and one part of oxygen we have as the composition of the after-damp formed by the mixture requisite for complete combustion, $CO_2 + 2\,(H_2O) + 16\,N$, thus:



Although for complete combustion about ten parts of air are required, a mixture of but five and a half parts of air to one part of gas will explode (but not violently), and a dilute mixture of as much as fourteen parts of air to one of gas may explode when exposed to a naked light.

Expressed in terms of the percentage of gas to the volume of the mixture, the minimum percentage of gas in explosive mixtures is about $6\frac{2}{3}$ *per cent.* and the maximum about $15\frac{1}{2}$ *per cent.*: when the mixture contains less than $6\frac{2}{3}$ per cent., there is not sufficient *gas*, and when it contains more than $15\frac{1}{2}$ per cent. there is not sufficient *oxygen* present to form an explosive compound.

The *after-damp* formed by the explosion of a mixture containing a *minimum* ($5\frac{1}{2}$ parts) *amount of air* may consist of an uncertain mixture thus:



The relative amounts of carbonic acid gas (if any), carbonic oxide, water, free hydrogen, (or deposited carbon?) have not been ascertained. If free hydrogen exists,—and we can hardly doubt that it does exist,—in such after-damps, the cause of *multiple explosions* is easily explained. An after-damp containing free hydrogen and carbonic oxide on becoming again mixed with air would form a highly explosive mixture.

Disregarding the possibility of the formation of small percentages of nitrogen compounds the after-damp from such a mixture ($5\frac{1}{2}$ parts of air) may contain from 5 to 0 parts of carbonic acid, from 0 to 5 parts of carbonic oxide, from 1 to 6 parts of water with free hydrogen (or deposited carbon?) and nitrogen; thus, on the supposition of free hydrogen, we have the series:

(1) $5\,CO_2$ . . + H_2O+18 H+44 N; or
(2) $4\,CO_2$+CO +2 H_2O+16 H+44 N; or
(3, 4, 5) . . . .
(6) . . 5 CO +6 H_2O+ 8 H+44 N;

or, on the supposition of no free hydrogen:
2 (CH_4)+$2CO_2$+CO+6 H_2O+44 N.

This seems to resemble the explanation given by Sir Humphrey Davy for, according to his view, the excess of marsh gas in such mixtures is left unaltered and the fire-damp would always contain for each volume of gas present:

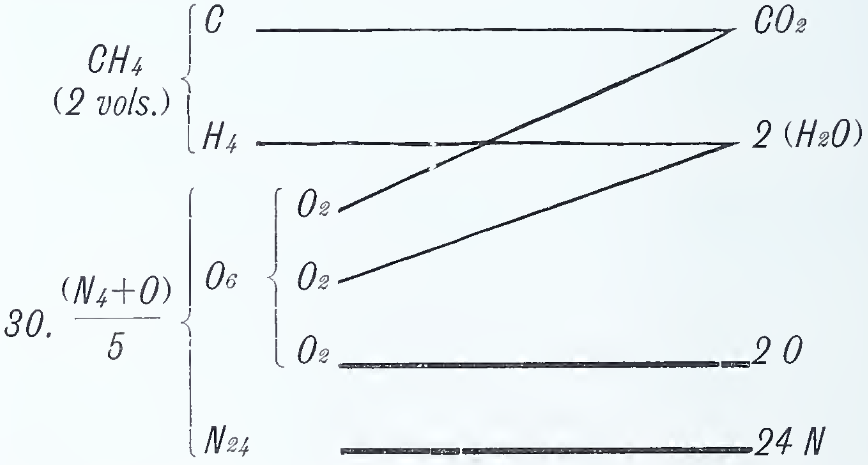
$CO_2+2\ (H_2\ O)+16\ N+$excess of CH_4.

but this theory has been shown to be incorrect.

The supposition of "deposited carbon," would permit the following formula: $CO+10\ H_2O+4\ C+44\ N$, but reactions of this class have not been obtained in experimental tests.

Until a series of careful experiments and analyses are made to determine the exact composition of the after-damp formed by different explosive mixtures, our knowledge of the subject cannot be advanced; theorizing here seems little better than guesswork.

When the mixture contains a surplus quantity of air (more than ten parts) all of the carbon will exist in the after-damp as carbonic acid gas and no explosive gas will remain, thus in a mixture containing 15 parts of air:



Fire-damp (light carbureted hydrogen CH_4) as it usually occurs in mines has been found to be mixed with a small percentage of carbonic acid gas and nitrogen. In some cases the blowers give off as much as *three per cent.* of the former and from *two to five per cent.* of the latter (Thomas).

Blowers. A blower is a strong discharge of gas from a crack, fissure, or cleavage joint in the coal, the floor, or the roof. They are very common in the mines of the Wyoming district and in some of the deeper mines of the Schuylkill region.

When small, they occasion no trouble other than the precautions necessary to insure the removal of the gas by the ventilating current, and to prevent its accumulation near the roof: But sometimes they are very powerful, and the amount of gas given off is so great that working must be suspended for a time. The great difficulty with large blowers is not to carry off the gas, for this may be done if the ventilation is sufficient, but to prevent them from catching fire, or rather to extinguish them when they have been fired.

Blowers of moderate size are often allowed to burn in preference to sending the explosive gas into the ventilating current, but very large blowers cannot be lighted without more or less risk of fire. As they inevitably catch fire sooner or later, it is necessary to provide some means of extinguishing them. This is accomplished by water, pipes being laid to convey the water from the sump in an upper level, or from the surface, to the working face. It is often found impossible to extinguish a blower when first lighted, even with two or three streams playing upon it with a strong head of water. It is then allowed to burn until the coal in its immediate vicinity has become very hot; on turning the water upon this coal, it is immediately vaporized, and the steam quickly extinguishes the flame.

Outbursts of gas.

An outburst of gas is an expression applied to any sudden increase in the amount of gas found in the air-ways or working places, and may result from an actual increase in the amount of gas being given off by the coal, or from an influx of gas which has accumulated in old workings.

This phrase is, however, not usually applied to an influx of gas from old workings when caused by a roof-fall or other accident.

The following analysis of fire-damp explosions shows that while the number of explosions directly attributable to barometric changes (causing outbursts of gas) is comparatively small, it has been shown that in Great Britain nearly all of the disasters resulting in great loss of life, are directly attributable to this cause.

An Analysis of the Fire-damp Explosions in the Anthracite Coal Mines, from 1870 to 1880.[*]

The table given below is compiled from the reports of the Inspectors of Mines, for the years from 1870 to 1879 inclusive. In it are included all recorded explosions, whether resulting in serious or trivial casualties. The majority of these were caused by the ignition of but a few cubic feet of explosive mixture, but some were terrible disasters, the victims of which may be numbered by scores.

The total number of recorded explosions is 639, the number of casualties 1127, and of these 225 resulted fatally. As the reports for two or three years are not complete, these figures do not represent the exact number of casualties ; but they express sufficiently well the ratio between the explosions and the number of miners injured by them.

During these years the explosion of fire-damp was the cause of sixteen per cent. of the total casualties reported by the inspectors, of eleven per cent. of the fatal accidents and of eighteen per cent. of the non-fatal casualties.

EXPLOSIONS OF FIRE-DAMP.

	1870.	1871.	1872.	1873.	1874.	1875.	1876.	1877.	1878.	1879.	Total.
January, ...	0	8	3	2	2	3	5	7	2	4	36
February, ...	2	4	4	2	5	3	6	4	1	5	36
March, ...	0	2	6	3	3	3	4	5	0	8	34
April, ...	4	7	14	6	5	2	7	8	2	10	65
May, ...	3	10	8	9	8	1	11	9	2	18	79
June, ...	1	6	7	13	12	5	4	6	4	3	61
July, ...	3	11	5	4	7	7	2	7	4	6	56
August, ...	3	10	13	10	5	3	2	3	6	9	64
September, ...	4	9	6	10	9	12	4	2	0	10	66
October, ...	8	7	10	6	8	6	8	7	2	10	72
November, ...	6	3	2	7	9	5	4	7	2	7	52
December, ...	6	6	3	2	4	9	8	6	4	10	58
Total, ...	40	83	81	74	77	59	65	71	29	100	679

Explosions in Eastern District of Luzerne not included in the report for 1871.
Explosions in Southern District of Carbon and Luzerne not included in column for 1875.
Shamokin District statistics omitted from report for 1876.

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[*] Enlarged from a paper read before the American Philosophical Society by the author, May 6, 1881.

The table is arranged to show the number of explosions occurring in each month of the year for ten years,—the right hand column shows the number for each month of the whole period.

An inspection of latter column shows at once that from April to October the number of explosions is far greater than for the remaining months of the year. In these seven months 463 explosions are recorded, an average of sixty-six for each month, but for the remaining five months (Jan., Feb., March, Nov., and Dec.,) we find but 216 explosions, an average of forty-three for each of these months.

Temporary or partial suspension of mining during some of these months in certain years may partly account for this difference, but is inadequate to explain so marked a contrast between the groups of warm and cold months.

It seems probable, that if a closer differentiation could be made, it would be found that many of the explosions occurring during the warm months, happened during or immediately following a short period of unusual warmth, during which the ventilating current was somewhat diminished in strength. At such a time, when unusual warmth with high barometer existed, a sudden fall in the barometric column, presaging a local or general storm would surely be accompanied by an increased outflow of gas which might readily become explosive at the working face while the air still remained safe in the upcast.

The low rate for July may be due in part to partial suspension of operations during that month, and the high rates of November and December, high compared to those of January, February and March, are probably due in part to steady working to supply the winter demand, and the low rates of the remaining winter months to partial suspension of work.

The maximum rate in May, and the next in rank, October, are just five months apart. Are these months subject to greater and more sudden and frequent barometric changes than others in this part of the United States?

A list of the most serious colliery disasters in Great Britain, from 1778 to 1866 inclusive develops the interesting fact that ont of forty-five explosions, ten occurred in June and eight in December, periods just six months apart.

The table is as follows:

Months.	*No. of Explosions.*
January,	2
February,	1
March,	4
April,	0
May,	4
June,	10
July,	4
August,	3
September,	2
October,	4
November,	3
December,	8
Total,	45

This list embraces explosions resulting in great loss of life only. The minimum loss of life was 20, and the maximum, the Oaks Mine disaster, December, 1866, was 362; the loss of life aggregates 2,621, an average of more than 58 for each disaster.

The occurrence of three of these explosions on June 2, 1862 (at Washington, Guindræth and Coppal,) and two of the most fatal on December 12 and 13, 1866 (the "Oaks" and "Talk o' the Hill," by which 362 and 92 lives were lost, certainly point to atmospheric disturbance as the immediate cause. The occurrence of a large percentage of these disasters at semi-annual periods, June and December, seems to indicate the occurrence (in Great Britian) during these months of marked atmospheric disturbances, as the probable cause of these great oubursts of gas.

But the problem I have been considering is somewhat different, for the table embraces *all* the explosions, whether large or small, occurring during the ten years. It shows a decidedly larger number for the warm than for the cold months, and therefore points primarily to *impairment of ventilation from high temperature*, rather than to barometric changes, as the true cause of the difference; but the occnrrence of two maximum periods,—May and October,—

seems to indicate that *barometric changes have also* exercised an important influence on the relative efflux of gas.

The *amount* of rise and fall does not seem to have a perceptible effect, for *the monthly barometric range is greatest during the cold months*, whereas fire-damp explosions are most frequent during the warm months. Frequent and *abrupt* changes from high to relatively low barometric pressure, are the probable cause of many explosions, although the movement of the mercury may not amount to more than one-eighth or one-quarter of an inch. *An unusually high barometric column* is always an intimation of coming or present danger, and a rising barometer should always be considered an indication of danger.

This is evidently the view now held by those in charge of the issue of colliery warnings in England, in connection with the signal service. The results attained in the past two years seem to indicate that something similar might be advantageously adopted in this country. These results for 1882 are summarized by the following extract from "Coal" republished January 31, 1883 from the *Colliery Guardian:*

"Thirty-two colliery warnings were issued during the year, nineteen of which were justified by subsequent events; twelve were followed within less than three days by fifteen explosions causing 139 deaths, two by forty-three deaths on the fifth day, and twenty-three lives were lost on the sixth day, or a total of 205 lives lost in six days from the issue of the warnings. Five lives were lost on the day of issue, the warnings being published in the press on the following day. In each case, those explosions which occurred on the fifth and sixth days were preceded by warnings indicating prolonged unfavorable conditions. Adding to our list four continental explosions, causing ninety-eight deaths, and tabulating them according to the atmospheric conditions existing at the time, we find that with a rising barometer there were twenty-three explosions, causing 298 deaths; with a falling barometer, six explosions, all non-fatal; and with a steady glass, one explosion, causing five deaths. Looking at the results in another way, there were twenty explosions, causing 227 deaths with the barometer ranging from 29.9

to 30.6 inches, the mean being 30.15 inches; with the barometer below 29.9 inches, there were ten explosions, causing seventy-six deaths, the mean barometer being 29.5 inches. Only ten out of the thirty disasters were accompanied by winds south of west and east.

"These facts support the belief, which is gradually gaining ground, that the period of greatest danger to miners is that when the barometer is high or rising, although fire-damp undoubtedly appears in greater quantities in the mine when there is a decrease of atmospheric pressure. Some well-known mining engineers declare that the gas makes its presence felt long before the barometer shows a sign of falling; that the gas, in fact, is so sensitive a barometer as to denote changes of pressure much earlier than mercury. It is highly probable that officials and men, being conversant with the fact of more gas appearing with decreasing pressure, relax their precautions against their unseen enemy when the mercury begins to rise. Unless the whole of the gas which has escaped during the falling of the mercury has been driven out of the mine, it follows that accumulations are forming in every available space, spaces which have probably been officially declared free of gas an hour or two before. It appears most desirable, therefore, that the additional care which is admittedly exercised with decreasing barometric pressure, should not in the least degree be relaxed when the mercury is rising; and when pressure is high, special search should be made for any accumulations, with a view to dispersing them before they become dangerous. By this means, the only gas to be dealt with when the barometer falls would be that escaping from the coal."

White-damp.

In addition to fire-damp, *white-damp* or carbonic oxide (CO) is occasionally present in anthracite collieries, being given off by the coal (?), by smoldering gob-fires, or existing as a part of the after-damp produced by an explosion of fire-damp, or from the explosion of powder.

Carbonic oxide (white-damp) is an exceedingly poisonous gas, acting almost instantly upon the system as a narcotic.

It is not readily detected when present in moderate quantities as it is colorless, nearly odorless, and does not affect or extinguish a lamp flame unless present in very large quantities. Its specific gravity is 0.968.

Its effects upon the system are much more marked than those of carbonic acid gas. Air containing a very large percentage (8 to 10) of the latter, may be breathed for a time without serious effects, but the inhalation of air containing a much smaller amount of carbonic oxide will speedily produce unconciousness (*coma*) resulting in death unless active restorative measures are immediately adopted.

It has been suggested that artificial respiration should be adopted in all cases of supposed death from the inhalation of this gas. After a fire-damp explosion it is not uncommon to find men apparently entirely uninjured by the explosion, but smothered by the after damp. When the body is found in an attitude indicating the entire absence of any dying struggle, it is highly probable that death (or coma ?) has resulted from the presence of white-damp (CO) and *not* from the presence of carbonic acid, or from a lack of oxygen.

In such cases a trial should be made to resuscitate by artificial respiration, and the administration of alcoholic stimulants, (brandy or whiskey,) even should there be no perceptible signs of life discernible on the closest examination.

White-damp is seldom found unmixed with other gases. When produced by a gob-fire, or by an explosion, it is always accompanied by more or less carbonic acid gas.

When mixed with free hydrogen and air (in after-damp) it forms a highly explosive mixture. Alone, or in presence of carbonic acid gas it is not combustible at normal temperature. It will not support combustion, but when present in even large proportions it does not extinguish a lamp.

The inhalation of air containing very small quantities (less than one per cent.) produces a very painful headache and general lassitude. If the inhalation be continued alarming symptoms and death may follow.

Black-damp.

Carbonic acid gas (CO^2), "black-damp," or "choke-damp" is found in greater or lesser quantities in all collieries. It is given off by many coals, either mixed with the fire-damp or separately, is produced by mine fires, by the gradual oxidation of gob in old workings, by the exhalations of men and mules, by lamps and fires underground, and by explosions of fire-damp.

Its specific gravity is 1.524, or about one and a half times as heavy as air; it therefore quickly sinks and settles into the lowest parts of the workings.

The thorough system of ventilation adopted in the anthracite regions generally keeps the working places and traveling-ways comparatively free from this gas; but in old workings it often accumulates in large quantities, and in mines containing smoldering gob-fires, or connecting with collieries on fire, this gas is often very troublesome.

As it lies along the floor, it is possible to enter old workings and gangways containing a very large quantity of this gas, the air in the upper part of the gangway containing only a small percentage.

As the inhalation of air containing eight or ten per cent. or more of carbonic acid gas can be continued for some time without producing serious effects, and as the presence of even smaller percentages is indicated by the feeble light given by a lamp or by its extinguishment, the danger of casualties from the presence of this gas is much less than from that of carbonic oxide.

Standing or walking in this gas occasions a feeling of numbness, or dull pains in the ankles, legs, and knees, followed by headache and sometimes by nausea.

Sulphuretted hydrogen.

Sulphydric acid, hydrogen sulphide, (H_2S) or sulphuretted hydrogen is formed by the decomposition of pyrites in the gob. It is seldom present in large quantity. The presence of a very small percentage of this gas can be detected by its disagreeable odor, (that of rotten eggs).

When inhaled in large quantities it acts as a powerful narcotic, but it is seldom present in quantities sufficient to produce this effect. Its specific gravity is 1.175; it does not support combustion, but is inflammable.

When mixed with oxygen it becomes explosive, and if mixed with fire-damp, free hydrogen, or white-damp, and the requisite amount of air, it doubtless increases the explosive force of the mixture.

Experiments have shown that it is very poisonous to some animals, but the same can hardly be said of its action on human beings. Air containing from one to three *per cent.* of this gas has been inhaled without any perceptible harmful effect, and chemists working in poorly ventilated laboratories are frequently surrounded by an atmosphere containing a much larger quantity. At the same time, it must be admitted that the continual inhalation of even very small quantities will be surely followed by some constitutional trouble.

This is also true of the continued inhalation of air containing carbonic acid gas. A case illustrating the effect of the latter came under my notice at the Kehley Run mine. During the time efforts were being made to smother the mine fire with carbonic acid gas, Thomas Tempest was engaged in stopping the crop-falls, and other openings, to prevent leakage of air into the mine and to confine the gas that was thrown in from the generating furnace.

He is about forty years of age, naturally healthy and robust. He had been employed in this way for two months when I saw him; his appearance denoted anæmia and general mal-nutrition. He stated that he was constantly exposed to the escaping gas (carbonic acid) and was often forced to inhale air containing a large percentage. He had lost ten pounds in weight, and had become subject to chills and frequent "flushes of heat," had lost strength, but had not been confined to bed.

The common means of detecting the presence of these different mine gases have already been stated, but there are some other methods that demand special recognition.

Sulphureted hydrogen,—H_2S,—by its strong odor,

White-damp, ... CO,—no trustworthy method,

Black-damp, ... CO_2,—feeble combustion of lamp flame, etc.
Fire-damp, ... CH_4,—cap above lamp flame, &c.

It is to be greatly regretted that we have no simple means of quickly detecting the presence of white-damp,—its presence is usually only detected by its dreadful effects.*

The method of detecting the presence, and determining the amount of fire-damp commonly known as "*Trying the gas*," with a lamp, is always accompanied by more or less danger, but no more simple means has as yet been devised.

Some miners think they can always *smell* the gas when present in notable amount, and they are doubtless right, but there are comparatively few men with sufficiently delicate olfactory organs to detect this gas by the odor of other gases with which it is mixed. Others think they can detect it by a feeling of dampness or coolness upon entering into a mixture of the gas, others tell its presence by the "tink" or sound of the gauze; but it is evident that none of these methods are trustworthy because they all depend upon the extreme delicacy of some sense,—smelling, feeling, hearing.

The lamp test is accurate enough for all practical purposes, but the risk of introducing a safety lamp into an explosive mixture makes it at best a very unsafe method.

Mr. Galloway has made a series of tests with different mixtures of fire-damp and air and as a result gives the following table, (Journal of the Royal Society, May, 1876.)

The lamp-wick after being carefully trimmed, was drawn down until the flame consisted only of a small blue hemisphere, $\frac{1}{8}$ inch high and $\frac{1}{4}$ inch in diameter at the base, with a conical speck of yellow at the middle near the top.

"A mixture of one part of marsh gas with sixteen parts of air (1 to 16) gave a voluminous, waving, spindle-shaped, blue cap $3\frac{3}{8}$ in. high.

"1 to 18—A similar cap, 2 inches high, which burned rather more steadily.

"1 to 20—A cap $1\frac{5}{16}$ inch high, with nearly parallel sides to about two thirds of its height, and drawn out to a point at the top. This cap was perfectly steady and more distinct than any of the others.

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*Thus the death of the explorers at the Kehley Run mine fire, &c., and in many other accidents, which in the absence of exact knowledge, are assumed to result either from black-damp or fire-damp.

"1 to 25—A conical cap $\frac{1}{2}$ to $\frac{5}{8}$ inch high.

"1 to 30—A conical cap $\frac{3}{8}$ inch high.

"1 to 40—A conical cap $\frac{3}{16}$ to $\frac{1}{4}$ inch high.

"1 to 50—An exceedingly faint cap $\frac{1}{8}$ inch high, the top having the appearance of having been broken off.

"1 to 60—It was hardly possible to distinguish anything above the small oil flame . . ."

It appears from these experiments that the minimum quantity of gas that can be detected is about 2 per cent., (1 to 50) and the indication given by this amount is very slight. Two and a half per cent. (1 to 40) gives a well marked cap; but when we consider that 6$\frac{2}{3}$ per cent. of gas makes an explosive mixture, that the fire boss or miner has only a margin of about 4 per cent. between a mixture giving no perceptible cap and an explosive mixture, that the percentage of gas varies rapidly with height, and that it may vary more than this marginal 4 per cent. in very short distances, it is evident that this is not a good means of detecting its presence,—but as it is the most simple method it will be with difficulty replaced by any other more complicated method.

An apparatus for the detection of fire-damp and the determination of the percentage of gas is described by Mr. George Frederick Ansell in a paper read before the North of England Institute of Mining Engineers, June 2, 1866.

The action of the apparatus depends upon the difference in the rate of diffusion and effusion,—the osmoses,—of gases differing in specific gravity. In one form the device consists of an India rubber balloon confined beneath a lever upon which it makes a gentle upward pressure. Any increase in the amount of fire-damp causes a rapid distension of the balloon which increases the upward pressure upon the lever, thereby completing an electric circuit connected with an alarum bell or indicator above ground. Several other forms of the devise are described by Mr. Ansell but they all depend upon the same principle,—in one a piston working in a cylinder, into which the gas finds its way through an unglazed piece of pottery, is used to complete the electric circuit. Mr. Ansell has also devised a modified form of aneroid barometer which may be carried in the hand from one part of the mine to another and which will show by the variations of the needle the relative amount of gas present. The case of the instrument is provided with an air-tight valve and a wedgewood pottery diaphragm, through which the gas passes by endosmoses, thus increasing the pressure which is immediately indicated by the needle.

The difficulties in the way of the practical use of any of these instruments will probably always exclude them from our mines. The balloon and piston instruments require an electric circuit, and this, as well as the apparatus itself, may be difficult to keep in perfect working order. The aneroid instrument would be worse than useless in the hands of an ordinary miner. It records only the *difference* in the percentage of gas present at the two localities compared.

The analysis of accidents * occurring from 1871 to 1881 shows very clearly that the explosion of fire-damp is not the greatest risk to which the miner is subjected; the condition of our anthracite collieries is in this respect probably equal to that of fiery mines in any other country, and it does not seem possible for the colliery owners and engineers to make any improvement in the method of ventilating which shall appreciably reduce the risk of accident.

The miners themselves, the inside laborers, and other employés, are directly responsible for the occurrence of a very large number of explosions.

Explosions of coal dust.

The results obtained by English and French scientists experimenting on the influence of coal dust (suspended in the air) upon explosive or otherwise non-explosive gases, seem to point to the following conclusions:

That the presence of coal dust may act in two ways:

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* See Chapter on Roof falls.

1. By directly aiding and taking part in the explosion, and,

2. By prolonging the flame of a powder blast otherwise too short to reach an explosive mixture of fire-damp and air situated at some little distance from where the shot is fired.

It seems rather doubtful whether coal dust is capable of forming an explosive mixture with pure air. Mr. Galloway believes that a certain otherwise non-explosive mixture of fire-damp and air —($\frac{1}{60}$ to $\frac{1}{112}$)—may become explosive when permeated with coal dust, and M. Virpelleaux who has devoted much time to this subject substantially endorses this conclusion.

But there are several considerations opposed to this view of the necessity of the presence of fire-damp; these may summarized thus:

1. Although admixtures of coal dust and air may not be readily inflammable (explosive) under ordinary conditions, it seems probable that when suddenly and violently set in vibration by a powder blast, an otherwise non-explosive mixture may become explosive.

2. It is a well-known fact that flour and other fine vegetable powders may cause violent explosions.

Explosions have occurred in some collieries, notably one at *Berandière* in 1877 when no fire-damp had been detected for long periods, (22 years,) and in a colliery at *Campagnac* an explosion occurred in 1875 although fire-damp had *never* been detected in the seam mined.

It is evident that the danger from this source is confined to comparatively dry mines, and is greater in dry than in wet weather.

Mr. Galloway quotes M. Vital as saying:

"'Very fine coal dust is a cause of danger in dry working places in which shots are fired; in well-ventilated workings it may of itself alone give rise to disasters; *in workings in which fire-damp exists it increases the chance* of explosion; and when an accident of this kind does occur, it aggravates the consequences.'"

But while these conditions are doubtless correct as regards the dust of bituminous coals, it is certainly questionable whether anthracite coal dust will form an explosive mixture with air alone under ordinary temperature and atmospheric pressure, or whether it will increase the explosive force of an explosive mixture.

It is not an infrequent sight to see lights used in a coal breaker where the dust is so thick that it almost entirely excludes the sunlight, yet we do not hear of explosions from this cause.

This does not, however, prove that explosions might not occur if the air was powerfully detonated by firing a blast, nor that a cloud of dust added to a mixture of fire-damp and air will not increase the risk of accident.

The air of dry mines cannot be kept free from dust, but the amount of dust may be greatly lessened by keeping the gangways as clean as possible.

The tables given in the following chapter show the number of fatal and non-fatal casualties from explosions of fire-damp from 1871 to 1880 inclusive to be 239 and 982,—a total of 1221 casualties. From 1875 to 1880 the tables are complete. Comparing the figures given in the tables for those years, with the production, we obtain the following:

Explosions of Fire-damp, 1875 to 1880.

YEAR.	Fatal accidents.	Tons of coal mined.	Tons mined for each life lost by explosion of CH_4.
1875,	18	19,712,472	1,095,137
1876,	29	18,501,011	637,966
1877,	12	20,828,179	1,735,682
1878,	23	17,605,262	765,446
1879,	29	26,142,689	901,472
1880,	14	23,437,242	1,674,088
1875 to 1880,	125	126,226,855	1,009,815

Explosions of Fire-damp, 1875 to 1880.

YEAR.	Non-fatal accidents.	Total accidents.	Tons mined for each person killed or injured by explosions.
1875,	82	100	197,125
1876,	116	145	127,593
1877,	119	131	158,994
1878,	34	57	308,864
1879,	130	159	164,419
1880,	80	94	249,332
1875 to 1880,	561	686	184,004

Chapter XXIII.

Roof-falls and other accidents.

The following set of tables showing an analysis of casualties in the anthracite regions from 1871 to 1881 is here reprinted* from the Transactions of the Institute of Mining Engineers to demonstrate the large percentage of accidents from falls of the roof–rock, and coal.

The tables have been compiled from the annual reports of the Inspectors of Mines.

As we cannot assume that every accident has been recorded by the inspectors ; as many casualties terminating fatally after the lapse of a considerable period are inevitably recorded as "non-fatal ;" and as clerical errors may have crept into the compilation, the figures of these tables cannot be considered absolutely correct, but the summary is sufficiently reliable to show the features I wish to differentiate.

Under the heading "Miscellaneous" are included all accidents not directly attributable to explosions of fire-damp, roof-falls or falls of coal. Prominent among the numerous causes included under this head are premature or delayed explosion of blasts, accidental explosion of powder or cartridges, accidents from mine cars, falls in shafts, machine accidents, sliding of loose coal in pitching breasts, suffocation by choke-damp, etc., comprising in all 48 per cent. of the total number of casualties.

Falls of coal, while not belonging to exactly the same class of casualties with roof-falls, have been included under this head. The general summary shows that 36 per cent. of the total number of casualties are directly traceable to falls of roof and coal, this percentage constituting 44 per cent. of the fatal and 32 per cent. of the non-fatal accidents, or in numbers, there were 979 fatal and 1,848 non-fatal accidents from this cause in the ten years, from 1871 to 1880, inclusive.

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*From a paper by the author, May, 1881.

In 1864 the accidents attributable to this class in the coal-fields of Great Britain reached 45 per cent. of the total number of casualties.

Roof-falls are common to all mines and cannot be entirely avoided; but they are especially occasioned by *thick seams*, by clods of shale or rock loosely adhering to the roof, and, when the roof is fair, by *inattention to the proper placing and renewal of props*, by badly located *shots spending their force upon the roof*, by the *vibration caused by shots* fired in adjoining or distant workings, by *driving breasts too wide*, by improper *removal* of *props*, and by *explosions of gas*. Falls of coal are similarly occasioned, but are most frequently the result of improper underholing or taking-up of bottom coal, allowing too large a mass of unsupported coal above the miner.

The majority of casualties from roof-falls, and nearly all those from falls of coal, occur at or near the working face, and are directly attributable to the carelessness or poor judgment of the miner, or to his reluctance to take the proper precautions and exercise the proper care, because these involve some addditional labor and trouble for which he will receive no pecuniary reward.

An extremely dangerous roof is occasionally met with,—notably at Pittston, where it is locally called the "black rock," etc. It is a dark carbonaceous slaty shale, perfectly hard and firm when first exposed, but it soon swells and softens from exposure to the atmosphere, and breaks off in large masses. A shale roof is seldom safe, but is particularly dangerous when but a few feet in thickness and loosely adhering to a firm sandstone roof, or, if irregularly bedded, in lens-shaped masses, or fissured by cleavage joints. A large mass of such material may hang for a long period, sustained by its adhesion to the firm sand-rock above, only to fall without warning when this adhesion is lessened by exposure to the atmosphere, or from the vibrations caused by the firing of shots.*

When the roof is in fair condition, falls are often occasioned by driving the working places too wide, or if they have originally been of proper width, by "skipping" the pillars, or by the pillar coal spalling off in wedge-shaped masses.

Attention to the proper placing, renewal, and removal of props is of prime importance, and a systematic and periodical inspection of the roof in all travelling ways and working places cannot be too strongly urged. This inspection should be intrusted to a thoroughly competent, experienced, and intelligent man only, invested with the necessary authority to enforce his orders, and who should be held personally responsible for all such casualties occurring within his jurisdiction. I venture to suggest this as a means of decreasing the number of casualties from roof-falls, feeling that in this respect nothing can be expected from the miner himself.

This inspection is generally one of the duties of the mine boss,—the Mines Ventilation Act specifically designates this as one of his chief duties,—but the mine boss at a large colliery rarely has sufficient time to perform this duty as it should be performed. The most he can accomplish is to warn the men from time to time to set additional props, or to pry down loose roof rock or coal, but he cannot remain and see that his instructions are carried out by the miner; nor can he personally examine the roof in all working places to determine for himself its exact condition. His knowledge of the condition of the roof is mainly derived from the miner.

Again, a roof perfectly safe when examined by the mine boss in the morning may (as mining progresses) become dangerous at any time during the day.

When the roof is not known to be perfectly secure, mining should certainly be suspended, and the men removed to a safe retreat during the firing of shots in adjoining and even distant working places, to avoid the risk occasioned by the vibration of the roof.

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*See reports of the Inspectors of Mines, 1878, p. 209, and 1879, p. 301, for illustrated examples of accidents resulting from similar conditions.

The tables show a total number of 2827 casualties resulting from falls of roof and coal; it would be interesting and instructive to know how many of these could have been avoided, had proper precautions been taken. This is a matter beyond the control of the mine inspector; his visits are necessarily made at considerable intervals, and although he may often be enabled to greatly diminish the danger from this cause, by ordering additional propping, etc., he cannot enforce the constant watchfulness and care necessary in every mine where the roof is unsafe. I am convinced that a large majority of these accidents is due to criminal carelessness on the part of the miner himself, to the recklessness that comes to all men inured to danger, and to the disinclination so frequently exhibited to undertake any additional labor that does not offer an immediate and positive remuneration.

The casualties from explosions of gas number 1221,—16 per cent. of the total number,—of which 239 resulted fatally.

The accidents included under the head "Miscellaneous" embrace 48 per cent. of the total number of casualties, aggregating in all 3838, of which 989 resulted fatally.

The total number of casualties recorded by the inspectors for the nine years is 7886, of which 2207, or about 29 per cent., resulted fatally.

The "summary of percentages,"—Table XII*a*—develops a remarkably uniform relation between the relative number of accidents attributable to roof-falls, explosions, and other causes, and also between the percentages of those terminating fatally each year. This is also evident in Tables XIII and XIII*a*.

Under the head "Miscellaneous" are included all accidents not directly attributable to explosions of fire-damp, roof-falls or falls of coal,—prominent among the numerous causes included under this head are, premature or delayed explosion of blasts, accidental explosions of powder or cartridges, accidents from mine cars, from being kicked by mules, falls in shafts, machine accidents, from sliding of loose coal in pitching breasts, etc., etc.,—embracing in all 48 per cent. of the total number of casualties.

TABLE I.

Fatal Accidents in 1871.

	Roof-falls.	*Explosions.*	*Miscellaneous.*	*Total.*
Pottsville District, ...	4	14	12	30
Ashland "	20	2	34	56
Shamokin "	18	9	16	43
Southern "	12	0	16	28
Middle "	15	1	37	53
	69	26	115	210

TABLE I*a*.

Non-fatal Accidents in 1871.

	Roof-falls.	*Explosions.*	*Miscellaneous.*	*Total.*
Pottsville District, ...	*84	*98	*115	118
Ashland "				168
Shamokin "				120
Southern "	17	1	18	36
Middle "	27	31	32	90
	†144	†149	†239	532

TABLE II.

Fatal Accidents in 1872.

	Roof-falls.	*Explosions.*	*Miscellaneous.*	*Total.*
Pottsville District, ...	36	14	41	91
Ashland " . . .				
Shamokin "				
Southern "	13	1	11	25
Middle "	15	8	17	40
Eastern "	36	5	26	67
	100	28	95	223

TABLE II*a*.

Non-fatal Accidents in 1872.

	Roof-falls.	*Explosions.*	*Miscellaneous.*	*Total.*
Pottsville District, ...	63	103	99	265
Ashland "				
Shamokin "				
Southern "	10	5	23	38
Middle " . . .	33	24	64	121
Eastern "	68	24	95	187
	111	53	182	346

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* January 1st to October 15th. † Totals only approximately correct.

TABLE III.

Fatal Accidents in 1873.

	Roof-falls.	*Explosions.*	*Miscellaneous.*	*Total.*
Pottsville District,	52	23	51	126
Ashland “				
Shamokin “				
Southern “	21	1	16	38
Middle “	11	6	29	46
Eastern “	32	3	19	54
	116	33	115	264

TABLE III*a*.

Non-fatal Accidents in 1873.

	Roof-falls.	*Explosions.*	*Miscellaneous.*	*Total.*
Pottsville District,	90	77	212	379
Ashland “				
Shamokin “				
Southern “	12	2	32	46
Middle “	21	14	56	91
Eastern “	77	19	73	169
	200	112	373	685

TABLE IV.

Fatal Accidents in 1874.

	Roof-falls.	*Explosions.*	*Miscellaneous.*	*Total.*
Pottsville District,	5	13	17	35
Ashland “	20	3	21	44
Shamokin “	13	1	12	26
Southern* “				
Middle “	17	9	31	57
Eastern “	39	1	29	69
	94	27	110	231

TABLE IV*a*.

Non-fatal Accidents in 1874.

	Roof-falls.	*Explosions.*	*Miscellaneous.*	*Total.*
Pottsville District,	22	38	40	100
Ashland “	32	7	56	95
Shamokin “	59	20	71	150
Southern* “				
Middle “	25	32	48	105
Eastern “	35	10	44	89
	173	107	259	539

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*No report from this district in 1874.

TABLE V.

Fatal Accidents in 1875.

	Roof-falls.	*Explosions.*	*Miscellaneous.*	*Total.*
Pottsville District, ...	11	7	10	28
Ashland "	11	0	15	26
Shamokin "	16	0	22	38
Southern "	11	2	8	21
Middle "	19	6	38	63
Eastern "	29	3	40	72
	97	18	133	248

TABLE V*a*.

Non-fatal Accidents in 1875.

	Roof-falls.	*Explosions.*	*Miscellaneous.*	*Total.*
Pottsville District, ...	17	28	43	88
Ashland "	50	15	49	114
Shamokin "	60	3	45	108
Southern "	41	3	33	77
Middle "	20	32	48	100
Eastern "	21	1	80	102
	209	82	298	589

TABLE VI.

Fatal Accidents in 1876.

	Roof-falls.	*Explosions.*	*Miscellaneous.*	*Total.*
Pottsville District, ...	5	10	13	28
Ashland "	9	1	17	27
Shamokin "	14	1	22	37
Southern "	18	4	15	37
Middle "	23	7	15	45
Eastern "	20	6	18	44
	89	29	100	218

TABLE VI*a*.

Non-fatal Accidents in 1876.

	Roof-falls.	*Explosions.*	*Miscellaneous.*	*Total.*
Pottsville District, ...	19	34	13	66
Ashland "	14	9	25	48
Shamokin "	21	13	17	61
Southern "	28	17	29	74
Middle "	18	22	47	87
Eastern "	44	21	55	120
	144	116	196	456

TABLE VII.

Fatal Accidents in 1877.

	Roof-falls.	*Explosions.*	*Miscellaneous.*	*Total.*
Pottsville District, ...	5	10	12	27
Ashland “	20	0	9	29
Shamokin “	17	0	11	28
Southern “ . . .	17	0	9	26
Middle “	25	1	12	38
Eastern “	31	1	8	40
	115	12	61	188

TABLE VIIa.

Non-fatal Accidents in 1877.

	Roof-falls.	*Explosions.*	*Miscellaneous*	*Total.*
Pottsville District, ...	21	44	24	89
Ashland “	27	7	24	58
Shamokin “	31	7	28	66
Southern “	20	9	31	61
Middle “	34	24	63	121
Eastern “	64	28	82	174
	197	119	252	568

TABLE VIII.

Fatal Accidents in 1878.

	Roof-falls.	*Explosions.*	*Miscellaneous.*	*Total.*
Pottsville District, ...	5	3	6	14
Ashland “	13	0	13	26
Shamokin “	16	11	20	47
Southern “	15	2	13	30
Middle “	14	7	15	36
Eastern “	16	0	18	34
	79	23	85	187

TABLE VIIIa.

Non-fatal Accidents in 1878.

	Roof-falls.	*Explosions.*	*Miscellaneous.*	*Total.*
Pottsville District, ...	4	11	15	30
Ashland “	40	7	42	89
Shamokin “	44	12	71	127
Southern “	43	2	25	70
Middle “	34	2	63	99
Eastern “	34	0	54	88
	199	34	270	503

TABLE IX.

Fatal Accidents in 1879.

	Roof-falls.	*Explosions.*	*Miscellaneous.*	*Total.*
Pottsville District, ...	10	5	9	24
Ashland "	21	3	19	43
Shamokin "	25	4	17	46
Southern "	15	2	8	25
Middle "	30	12	23	65
Eastern "	30	3	26	59
	131	29	102	262

TABLE IX*a*.

Non-fatal Accidents in 1879.

	Roof-falls.	*Explosions.*	*Miscellaneous.*	*Total.*
Pottsville District, ...	45	35	78	158
Ashland "	40	18	53	111
Shamokin "	22	15	66	103
Southern "	44	8	48	100
Middle "	53	45	87	185
Eastern "	39	9	86	134
	243	130	418	791

TABLE X.

Fatal Accidents in 1880.

	Roof-falls.	*Explosions.*	*Miscellaneous.*	*Total.*
Pottsville District, ...	6	4	5	15
Ashland "	23	1	15	39
Shamokin "	15	8	11	34
Southern "	13	0	13	26
Middle "	7	1	17	25
Eastern "	25	0	12	37
	89	14	73	176

TABLE X*a*.

Non-fatal Accidents in 1880.

	Roof-falls.	*Explosions.*	*Miscellaneous.*	*Total.*
Pottsville District, ...	57	12	60	129
Ashland "	34	11	46	91
Shamokin "	37	36	51	124
Southern "	39	1	53	93
Middle "	31	20	80	131
Eastern* "	30	0	72	102
	228	80	362	670

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* Does not include very slight injuries.

TABLE XI.

Fatal Accidents 1871 *to* 1880.

Years.	*Roof-falls.*	*Explosions.*	*Miscellaneous.*	*Total.*
1871,	69	26	115	210
1872,	100	28	95	223
1873,	116	33	115	264
1874,	94	27	110	231
1875,	97	18	133	218
1876,	89	29	100	218
1877,	115	12	61	188
1878,	79	23	85	187
1879,	131	29	102	262
1880,	89	14	73	176
1871 to 1880,	979	239	989	2267

TABLE XI*a*.

Non-fatal Accidents 1871 *to* 1880.

Years.	*Roof-falls.*	*Explosions.*	*Miscellaneous.*	*Total.*
1871,	144	149	239	532
1872,	111	53	182	346
1873,	200	112	373	685
1874,	173	107	259	539
1875,	209	82	298	589
1876,	144	116	196	456
1877,	197	119	252	468
1878,	199	34	270	503
1879,	243	130	418	791
1880,	228	80	362	670
1871 to 1880,	1848	982	2849	5679

TABLE XII.

General Summary, 1871 to 1880.

Years.	*Roof-falls.*	*Explosions.*	*Miscellaneous.*	*Total.*
1871,	213	175	354	742
1872,	211	81	277	569
1873,	316	145	488	949
1874,	267	134	369	770
1875,	306	100	431	837
1876,	233	145	296	674
1877,	312	131	313	756
1878,	278	57	355	690
1879,	374	159	520	1053
1880,	317	94	435	846
1871 to 1880,	2827	1221	3838	7886

TABLE XII*a*

Summary showing Percentages.

	Roof-falls.	*Explosions.*	*Miscellaneous.*	
	Fatal. Non-fatal.	*Fatal. Non-fatal.*	*Fatal. Non-fatal.*	*Total.*
1871, . .	.10+.19=(.29)	.03+.20=(.23)	.16+.32=(.48)	=100 per cent.
1872, . .	.17+.20=(.37)	.05+.09=(.14)	.16+.33=(.49)	=100 “
1873, . .	.12+.21=(.33)	.03+.13=(.16)	.12+.39=(.51)	=100 “
1874, . .	.12+.23=(.35)	.03+.14=(.17)	.14+.34=(.48)	=100 “
1875, . .	.11+.26=(.37)	.02+.10=(.12)	.16+.35=(.51)	=100 “
1876, .	.13+.21=(.34)	.04+.18=(.22)	.15+.29=(.44)	=100 “
1877, .	.15+.26=(.41)	.02+.16=(.18)	.08+.33=(.41)	=100 “
1878, .	.11+.29=(.40)	.03+.09=(.12)	.13+.38=(.51)	=100 “
1879, .	.12+.23=(.35)	.03+.13=(.16)	.10+.39=(.49)	=100 “
1880, . .	.11+.27=(.38)	.02+.09=(.11)	.08+.43=(.51)	=100 “
1871–1880,	.12+.24=(.36)	.03+.13=(.16)	.13+.35=(.48)	=100 “

TABLE XIII.

Summary showing percentages,—fatal accidents.

	Roof-falls.	*Explosions.*	*Miscellaneous.*	*Total.*
1871, ...	.33	.12	.55	100 per cent.
1872, ...	.45	.13	.42	“
1873, ...	.44	.12	.44	“
1874, ...	.41	.12	.47	“
1875, ...	.39	.07	.54	“
1876, ...	.41	.13	.46	“
1877, ...	.61	.06	.33	“
1878, ...	.42	.13	.45	“
1879, ...	.50	.11	.39	“
1880, ...	.50	.08	.42	“
1871 to 1880, ...	.44	.11	.45	100 “

TABLE XIII*a*.

Summary showing percentages,—non-fatal accidents.

	Roof-falls.	*Explosions.*	*Miscellaneous.*	*Total.*
1871, ...	.27	.28	.45	100 per cent.
1872, ...	.32	.16	.52	“
1873, ...	.29	.17	.54	“
1874, ...	.32	.20	.48	“
1875, ...	.36	.14	.50	“
1876, ...	.31	.26	.43	“
1877, ...	.35	.21	.44	“
1878, ...	.40	.07	.53	“
1879, ...	.31	.16	.53	“
1880, ...	.34	.12	.54	“
1871 to 1880, ...	.32	.18	.50	100 “

TABLE XIV.

Pottsville district, 1874 to 1879.

Fatal.	*Roof-falls.*	*Explosions.*	*Miscellaneous.*	*Total.*
1874,	5	13	17	35
1875,	11	7	10	28
1876,	5	10	13	28
1877,	5	10	12	27
1878,	5	3	6	14
1879,	10	5	9	24
1874 to 1879,	41	48	67	156
Non-fatal				
1874,	22	38	40	100
1875,	17	28	43	88
1876,	19	34	13	66
1877,	21	44	24	89
1878,	4	11	15	30
1879,	45	35	78	158
1874 to 1879,	128	190	213	531
Total casualties,	169	238	280	687

TABLE XV.

Ashland district, 1874 to 1879.

Fatal.	*Roof-falls.*	*Explosions.*	*Miscellaneous.*	*Total.*
1874,	20	3	21	44
1875,	11	0	15	26
1876,	9	1	17	27
1877,	20	0	9	29
1878,	13	0	13	26
1879,	21	3	19	43
1874 to 1879,	94	7	94	195
Non-fatal.				
1874,	32	7	56	95
1875,	50	15	49	114
1876,	14	9	25	48
1877,	27	7	24	58
1878,	40	7	42	89
1879,	40	18	53	111
1874 to 1879,	203	63	249	515
Total casualties,	297	70	343	710

TABLE XVI.

Shamokin district, 1874 to 1879.

Fatal.	*Roof-falls.*	*Explosions.*	*Miscellaneous.*	*Total.*
1874,	13	1	12	26
1875,	16	0	22	38
1876,	14	1	22	37
1877,	17	0	11	28
1878,	16	11	20	47
1879,	25	4	17	46
1874 to 1879,	101	17	104	222
Non-fatal.				
1874,	59	20	71	150
1875,	60	3	45	108
1876,	21	13	27	61
1877,	31	7	28	66
1878,	44	12	71	127
1879,	22	15	66	103
1874 to 1879,	237	70	308	615
Total casualties,	338	87	412	837

TABLE XVII.

Southern district of Carbon and Luzerne, 1871 to 1879.

Fatal.	*Roof-falls.*	*Explosions.*	*Miscellaneous.*	*Total.*
1871,	12	0	16	28
1872,	13	1	11	25
1873,	21	1	16	38
1874,	—	—	—	—
1875,	11	2	8	21
1876,	18	4	15	37
1877,	17	0	9	26
1878,	15	2	13	30
1879,	15	2	8	25
1871 to 1879,	122	12	96	230
Non-fatal.				
1871,	17	1	18	36
1872,	10	5	23	38
1873,	12	2	32	46
1874,	—	—	—	—
1875,	41	3	33	77
1876,	28	17	29	74
1877,	20	9	31	60
1878,	43	2	25	70
1879,	44	8	48	100
1871 to 1879,	215	47	239	501
Total casualties,	337	59	335	731

TABLE XVIII.

Middle district of Luzerne and Carbon, 1871 to 1879.

Fatal.	*Roof-falls.*	*Explosions.*	*Miscellaneous.*	*Total.*
1871,	15	1	37	53
1872,	15	8	17	40
1873,	11	6	29	46
1874,	17	9	31	57
1875,	19	6	38	63
1876,	23	7	15	45
1877,	25	1	12	38
1878,	14	7	15	36
1879,	30	12	23	65
1871 to 1879,	169	57	217	443
Non-fatal.				
1871,	27	31	32	90
1872,	33	24	64	121
1873,	21	14	56	91
1874,	25	32	48	105
1875,	20	32	48	100
1876,	18	22	47	87
1877,	34	24	63	121
1878,	34	2	63	99
1879,	53	45	87	185
1871 to 1879,	265	226	508	999
Total casualties,	434	283	725	1442

TABLE XIX.

Eastern district Luzerne, &c., 1872 to 1879.

Fatal.	*Roof-falls.*	*Explosions.*	*Miscellaneous.*	*Total.*
1872,	36	5	26	67
1873,	32	3	19	54
1874,	39	1	29	69
1875,	29	3	40	72
1876,	20	6	18	44
1877,	31	1	8	40
1878,	16		18	34
1879,	30	3	26	59
1872 to 1879,	232	22	184	439
Non-fatal.				
1872,	68	24	95	187
1873,	77	19	73	169
1874,	35	10	44	89
1875,	31	1	80	102
1776,	44	21	55	120
1877,	64	28	82	174
1878,	34		54	88
1879,	39	9	86	134
1872 to 1879,	382	112	569	1063
Total casualties,	615	134	753	1502

TABLE XX.

Table showing percentages in the different districts, calculated from the total casualties of each district,—both fatal and non-fatal. 1871 to 1879.

	Roof-falls.	*Explos's.*	*Miscel's.*		*Total.*	*Fatal.*	*Non-fatal.*
Pottsville District,* .	.25	.34	.41	=	100%	.23	.77
Ashland " * . .	.42	.10	.48	=	"	.28	.72
Shamokin " * . .	.40	.11	.49	=	"	.27	.73
Southern " . .	.46	.08	.46	=	"	.33	.67
Middle " . .	.30	.20	.50	=	"	.31	.69
Eastern " .	.41	.09	.50	=	"	.29	.71
In all Districts, . .	.36	.16	.48	=	100%	.29	.71

The number of tons of coal sent to market for each life lost is shown by the following table. As the statistics prior to 1875 are not complete, it is not possible to extend this table further back than that date.

The tonnage sent out per life lost increases with the output (in much the same way that the cost of mining decreases); it will at the same time be observed that there are marked exceptions to this law.

Averaging the figures given for six years, we obtain 98,692 tons as the output *sent to market* for each life lost, equivalent to an *actual production* of about one hundred thousand tons per life lost, (allowing 1,302 tons for consumption at the colliery).

When we consider the *great thickness of the coal beds* now worked in the anthracite regions, the *steep inclination* of the beds, and remember that *accidents occurring in the breaker*, (a cause of accidents not found in other coal mining districts), *are included in these statistics* as well as accidents occurring on the culm and rock dumps, etc., we are almost persuaded that the mining discipline and the precautions taken to prevent accidents are equal if not superior to those of any European mining district.

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*The percentages of these districts cover the years from 1874 to 1879, inclusive.

Tons of coal shipped per life lost.

YEAR.	Fatal accidents.	Tonnage.	Tons mined per life lost.
1875,	248	19,712,472	79,486
1876,	218	18,501,011	84,867
1877,	188	20,828,179	110,788
1878,	187	17,605,262	94,146
1879,	262	26,142,689	99,781
1880,	176	23,437,242	133,166
1875 to 1880,	1,279	126,226,855	98,692

Again, this amount (100,000 tons) per life lost, does not by any means represent the amount of coal taken from the mine. It will be safe to assume that one-third of the material raised is discarded as worthless (dirt or fine coal and bony) in the process of preparation; therefore the 100,000 tons shipped represent nearly or quite *150,000 tons mined for each life lost*, and in certain years the quantity raised has been fully 225,000 tons.

In comparing the results we have obtained in the anthracite region with those of European districts where such a thing as "breaker waste" is unknown and the fine coal is included in the output* it would be eminently unfair to disregard the large amount of material mined and sent to the surface that is not now marketable and that is not included in the statistics of production.

"Mr. Evans, one of Her Majesty's Inspector of Mines, says in his report for 1875 that during the twenty years ending December, 1875, the percentage of deaths was one death for 127,740 tons of coal raised, and in 1875 one for 108,918 tons."†

Hyslop gives the average for twelve years ending in 1867 (in Great Britain) as one death for 83,830 tons of coal raised, and then gives the following later figures for comparison.

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* Being either burnt as such, coked, or made into artificial fuel.

† Bagot's "Accidents in Mines," page 98. This average seems very high.

1871,	1 death for 109,246 tons.
1872,	1 " " 116,409 tons.
1873,	1 " " 133,677 tons.
1874,	1 " " 133,251 tons.

to which we may add from Bagot :

1875,	1 death for 108,918 tons.

but Bagot's figures for 1873 and 1874 do not correspond with those given by Hyslop; thus he gives:

1871,	1 death for 109,246 tons.
1872,	1 " " 116,409 tons.
1873,	1 " " 129,843 tons.
1874,	1 " " 119,521 tons.

In later years the results have been much better; thus we find in

1881,	1 death for 177,106 tons.
1882,	1 " " 152,161 tons.

The figures I have compiled for 1875 to 1880, when compared with the above data, making allowance for the amount of waste not recorded, and for accidents occurring in the breaker, give us little cause to feel chagrined at the record they present.

A method of timbering up roof-falls on a gangway or airway is shown by Fig. 58 from a sketch made by Mr. Winslow at Prospect colliery.

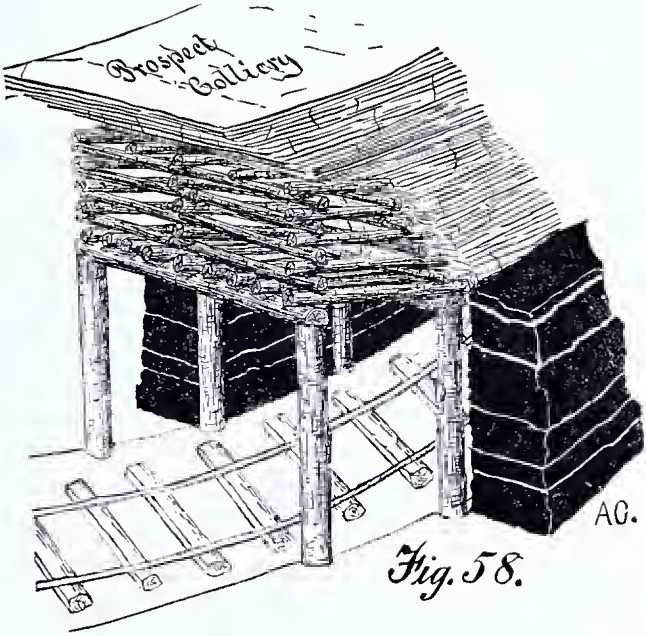

Fig. 58.

When there is no danger of more material falling this plan is not adopted. If the fall occurs on an airway and there is danger that gas may accumulate in the vacant space, a slant battery may be erected to drive out the gas.

Such a battery is shown by Page plate No. 40, taken from an illustration in one of the mine inspectors' reports.

CHAPTER XXIV.

Mine Fires.

It is often impossible to discover the origin of a mine fire. Fires sometimes originate in a way that leaves no doubt as to their origin, but a large number of fires have occurred under circumstances that have buried in impenetrable mystery the manner in which they originated.

Accidental firing of the stables, or the smithy, the ventilating furnace*, an unusually strong blower of gas, fire in engine rooms underground, and carelessness of the miners in leaving burning material in the working places are some of the known causes. Mine fires have also originated from fires lighted at the outcrop, from burning culm heaps, and from fire in crop-falls.

The origin of a large number of fires, especially those occurring in old workings, has been attributed to the spontaneous combustion of gob left in the breasts; but comparatively few engineers now countenance this view.

Of the spontaneous combustion of bituminous slack coal there can be no doubt; likewise, there seems to be little doubt that hard-dry anthracite will not fire spontaneously, but many believe the softer anthracites found in the western portions of the anthracite basins will fire spontaneously.

Those who take the latter view, point to the large number of burning culm dumps in the western parts of the different basins, assuming that these have (many of them) taken fire spontaneously.

Those who dissent from this opinion, see in these burning heaps only a demonstration of the fact that the coal they contain is more easily ignited than the harder varieties, and believe that the fires have originated from locomotive sparks, fires built by tramps, or by poor people engaged in picking coal from the heaps, or in some other accidental way.

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*Now fortunately in use at few mines.

The heat developed by decomposing pyrites in large culm banks is certainly very great, but whether it is sufficient to fire anthracite coal has as yet neither been proven nor disproven.

When this heat reaches a certain point, the coal and coal slates will give off a considerable percentage of gas. If this gas comes in contact with fire at the surface of the dump, (burning cotton-waste, for instance,) we can readily conceive how this fire might quickly spread to the center of the heap and ignite that part already heated.

The theory of spontaneous combustion has also frequently been advanced as an explanation of the origin of mine fires.

If this theory is correct, if anthracite culm and gob will take fire spontaneously, it is important to avoid leaving large heaps of gob in the workings.

The reason why this theory has been so frequently advanced, is attributable to the fact that many mine fires, when first discovered, have been confined to the gob in old workings, but it does not follow that, because the gob is burning and no miner can be found who will acknowledge having any connection with the origin of the fire, that its origin was spontaneous.

When gob takes fire from any cause, the fire may smoulder for a long time and spread until a very large quantity of material is ignited before the fire appears at the surface of the heap. This may be due in part to the direction of the draught through the heap, and in part to the large amount of slate and rock contained in the mass.

The precautions to be taken to prevent mine fires are obvious. To prevent their occurrence in old workings, it is absolutely necessary to keep the employés from entering them, but notwithstanding rules to the contrary this cannot always be done.

Next to prevention, the early discovery of a fire is the most important measure. To accomplish this a periodic inspection of all the old workings that can be entered, is made by the mine or fire-boss, and sometimes by the superintendent.

A smouldering fire is usually detected before it has gained much headway by the carbonic acid gas sulphurous acid, and heat in the upcast air, but when the fire is in old unventilated workings it often reaches unmanageable proportions before its existence is suspected.

The means now successfully used in extinguishing are:

1. By throwing water directly on the burning gob or coal;
2. By drawing the burning material (gob) out of the breast;
3. By sealing the mine, or portion on fire, to exclude the air, and
4. By sealing and introducing steam;
5. By flooding with water.

As flooding is often a very expensive process, involving the building of very massive dams, etc., and as it is ruinous to the mine, rotting the timbers, ruining the road-beds, weakening the roof, and occasioning serious roof-falls, it is only resorted to after all other means have failed, or when the condition of the colliery is such that no other method offers any prospect of success.

Drawing the burning or heated material from a breast can only be accomplished in steep-pitching breasts and when the amount of stuff on fire is comparatively small, and is objectionable because while the drawing process is being carried on the air cannot be excluded.

When the portion on fire communicates with workings open to the surface through crop-falls, or is located in workings directly beneath the outcrop, it is impossible to prevent the entrance of air.

This has been attempted at many collieries, but the most that can be accomplished is to prevent the fire from spreading rapidly,—it cannot be smothered in this way.

The plan usually adopted is to tamp all the vents that can be found with fine clay, earth free from stones, or rotten outcrop coal dirt ("smut.")

The introduction of carbonic acid gas has been unsuccessfully tried at several places. Its failure may be attributed

1. To the impossibility of excluding the air,—in other words to the impossibility of making a mine, open to the surface through crop-falls, even approximately air-tight, and

2. When the gas is generated by a furnace, to the high temperature of the injected gas, and to the possible production of carbonic oxide by the generating furnace.

When the mine, or that portion on fire can be sealed up so that only a small quantity of air can find its way into the seat of the fire, the carbonic acid process might be a valuable adjunct to the method of smothering, either by sealing or by sealing and injecting steam.

It is evident that the carbonic acid process can only be successfully used when the amount of gas injected is in excess of the capacity of the leaks (vents).

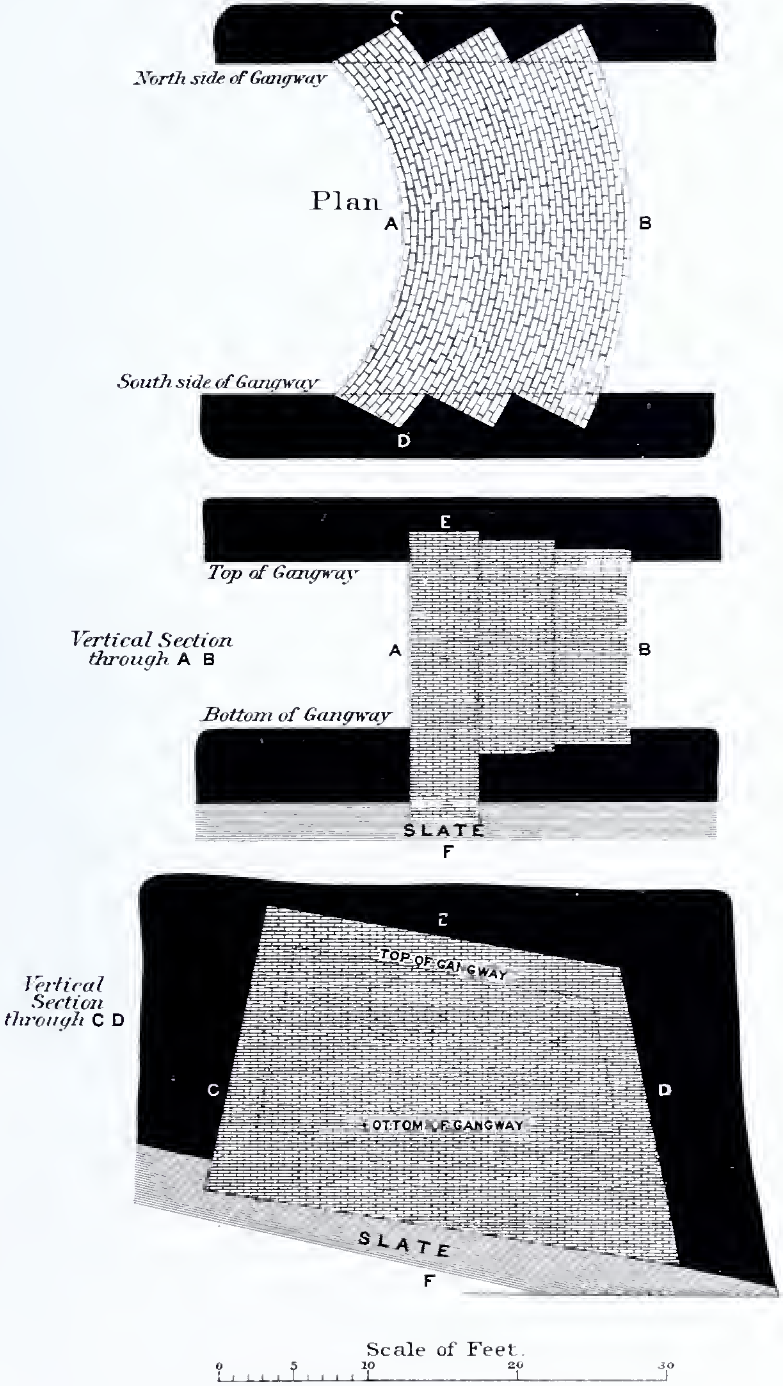
But when the mine or that portion on fire can be thoroughly sealed so that it is approximately air-tight, the amount of carbonic acid gas given off by the fire itself will probably accomplish all that could be effected by the injected gas.

When the fire is discovered before it has made much headway, an effort is generally made,—when at all practicable.—to extinguish it by streams of water thrown directly upon the burning coal and gob.

To protect the men from the steam, heat, and carbonic acid gas it is necessary to direct a strong ventilating current of air through the workings,—this is one of the greatest objections to this method.

When every method fails and the fire is located above the water line and cannot be flooded, there is still one resort left by which the (fire which cannot be extinguished) may be prevented from spreading into untouched, coal or into adjoining workings. This method involves the removal of a strip of coal of such width that the fire cannot spread across the interval. This is termed "cutting out the coal," and is sometimes followed by filling in the cut so made with rock and clay or earth, or better still, earth or clay free from fragments of rock. The enormous expense attending this

BRICK DAMS BUILT TO FLOOD KEHLEY RUN COLLIERY.



Designed by Heber S. Thompson, M.E.

method of confining or isolating a mine fire in a thick coal bed like the Mammoth, prevents its general adoption.

When a mine fire extends to solid coal its further progress is practically arrested for the time being. A solid face of anthracite coal burns very slowly even when well supplied with air; but the partings of carbonaceous slate or shale, and top slates, burn much more rapidly. Collieries that have been on fire often show that while the fire has extended but a few inches into the pillar coal, it has penetrated to a depth of two feet or more into the parting slates,—it therefore follows that a fire will make more rapid progress in a bed of coal containing several partings of carbonaceous shale or slate than in a solid seam of coal.

The explanation of this peculiarity will probably be found in the fact that the heat warps, shrinks, and cracks the slaty or shaly partings, allowing the air to penetrate to a greater depth.

The actual cost of extinguishing some of the principal mine fires that have occurred in the anthracite regions has ranged from twenty up to one hundred and fifty thousand dollars. This does not include the loss from permanent damage to the mine, the cost of repairs necessary to place the mine in working order, nor the loss occasioned by suspension of mining.

Page plate No. 41 shows a sketch of the brick dams built at the Kehley Run colliery to prevent the flooding of the Kohinoor workings. The gangway was eighteen feet wide and ten feet high. The dams were built in three sections with a radius of fifteen feet, with the convex side towards the area to be flooded. The sections were each made five feet thick. "Nearly two hundred thousand brick and six hundred barrels of cement . . . . were used in their construction." A copiously illustrated description of the methods employed is given in Mr. Heber S. Thompson's report to the Directors of City (Philadelphia) Trusts for 1881, the main features of which may also be found republished in the Report of the Inspectors of Mines for 1881.

CHAPTER XXV.

Hygiene of Mines.

Before proceeding to a discussion of the diseases most common among miners, it will not be amiss to quote here from Dr. Bucks' work on "Hygiene and Public Health" a description of the miner's daily life and surroundings found in a chapter contributed to that work by Mr. H. C. Sheafer.

Mr. Sheafer gives us an excellent pen picture, drawn, however, from a standpoint from which the shady side only seems to have been visible.

I have quoted below those parts of his article which seem to me most likely to furnish those unfamiliar with anthracite mining, a true picture of the miner's daily life, the risks he assumes, and the hardships he must endure.

"The working miner usually devotes his whole life to that occupation. He frequently, perhaps generally, begins at the age of from eight to twelve years as a slate-picker in the breaker—the building in which the coal is prepared for market—where his business is to sit all day, with twenty or thirty [or more] companions of about his own age, and pick out fragments of slate from a thin stream of coal constantly flowing past him. The place in which he works is apt to be more or less open and exposed to draughts. His clothing consists of shirt and pantaloons, usually old and ragged; a battered cap and a pair of coarse shoes—the last often omitted in summer. His whole costume, whatever its original color, is soon stained a uniform black by the thick cloud of coal-dust which fills the breaker, filters through his clothing and begrimes his skin, and which forms a large component part of the atmosphere he breathes. As boy and man, his invariable practice at the close of every working day is to wash himself thoroughly from head to foot, a custom to which his hardiness and generally rugged health in early life are to be largely attributed. His diet, as boy and man, is simple. Pork, salt fish, potatoes, and home-made bread are its staple constituents; but when work is good and money sufficient, all the luxuries of the local market are to be found on the miner's table. He learns to smoke and chew tobacco at an early age, has few or no scruples against the use of either malt or alcoholic liquors, and withal grows up to be a lusty, sinewy youth, who seldom troubles the doctors, unless overtaken by one of the numerous accidents to which his own recklessness not less than his somewhat dangerous occupation exposes him. [His early life may be spent inside the mine, first as door-tender, then as driver, etc.] At the age of eighteen or twenty, if he has not previously entered the mine as a driver, or for some other description of boys' work, he goes in as a laborer, becoming in effect, though not in name, an apprentice to a practical miner, with duties so nearly the same as those of his boss, that, for the purposes of this article, they may be considered identical.

"The miner gets to his work shortly before seven o'clock in the morning, if on the day shift, or between five and six in the evening, if on the night shift*. He is dressed in flannel shirt, woollen or heavy duck pantaloons, heavy shoes or boots, and usually with a coat thrown loosely over his shoulders. On his head he wears a cap, a slouch hat, or a helmet shaped like a fireman's, but of smaller dimensions. Whatever the headgear, his lamp, a small tin one shaped like a miniature coffee-pot, swings by a hook over the visor; unless the place in which he works is fiery, when he carries a safety-lamp in his hand. His dinner-can and canteen of water or cold tea are swung from a strap passing over his shoulders. Thus equipped he rides down the shaft or the slope, and, if he is lucky enough to catch a train of empty mine-wagons going to his working-place, he rides in, a distance, it may be, of two or three miles from the foot of the shaft. If no wagons are at hand, he walks most of the way through water and slush, taking small account of wet feet, or indeed of wet clothing at any time, though the roof over him may drip all day long. It is an exceptional case if he wears a rubber or oil-cloth suit, even in the wettest places. [However, wet road-beds are almost unknown in some collieries.]

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* Night shifts are seldom worked except in gangway or tunnel driving, working on second outlet, etc.

"Two miners, or two miners and a laborer, form a gang, and their work is an alternation of exhausting physical labor and intervals of rest. They work with drilling-bar, powder, and pick, getting down the coal and breaking it to a size small enough to handle; with drills, preparing and charging a hole for blasting; with shovels, clearing away the coal and getting it into the mine-cars to be sent to the surface; and then, when a particular job is done, or a blast is to be fired, they repair to the nearest place of safety, and in their overheated condition sit down in the cold, damp draught of the ventilating current to cool off as rapidly as possible. In walking to and from his work along the mine gangway, the miner tries to step on the sills on which the track is laid, thus avoiding the hollows worn by the mules' feet between the sills; and as these are laid from two and a half to three and a half feet apart, the effort gives him a long, slow, swinging gait,* the head being thrown forward to counterbalance the body. The same posture is found best for traversing the manways and other smaller passages, the long stride being advantageous in picking the way over rough and uncertain ground, while the bent head escapes projections of the roof, and permits the light of the lamp in the miner's cap to fall on the ground at his feet. The habit becomes fixed, and the old miner may always be known by his bent shoulders and swinging stride

"Among the most laborious of the miner's duties is setting the timbers which support the roof.

"The gangway timbers, unless the rock and coal are unusually solid, consist of a prop on each side, with a cross-piece uniting them. They are from 10 to 15 inches thick, of length adapted to the dimensions of the gangway, and being of green wood, are correspondingly heavy, weighing from 300 to 500 pounds, according to size. Yet three men are not only expected to set the side-pieces, but to lift the heavy cross-beam into position far above their heads, and fix it there. The work is so hard, performed as it is beyond the brattice which supplies fresh air, in an atmosphere more or less charged with powder-smoke and carbonic acid gas, that by the time it is done, all three are thoroughly exhausted and overheated, and in most favorable condition for the reception of colds, lung disorders, and rheumatism. If working in a steeply pitching breast,* though the timbers used are not so large, they are quite large enough to tax the strength of the two men who have to get them up a steep and difficult 'manway' by sheer lifting and pulling. In this way, which is almost like working up through a chimney, timbers averaging perhaps eight feet long by six inches thick are carried to the top of the breast, which may be from sixty to eighty yards above the gangway level.

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* This stride may be adopted without any attempt to walk on the sills, an effort rarely made unless the road bed is very wet.—H. M. C.

"One great cause of impurity in the atmosphere in which the miner works is that the brattice is frequently neglected, and the work pushed so far beyond it that it ceases almost entirely to affect the air at the face, which then becomes loaded with powder-smoke and carbonic acid, or, in fiery mines, carburetted hydrogen.

"Other portions (of the miner's work) consist of straining at arm's length to dislodge a mass hanging from the roof, of lifting and tugging at heavy weights, of shovelling continuously, hour after hour (where coal has to be shovelled into the mine-cars,) and of swinging a heavy sledge in drilling by hand-power. His footing is frequently unsteady, having to be maintained on a steep-pitching floor of smooth slate, so that, as a miner once expressed it to a friend of the writer, 'it is very much like asking a man to stand on the roof of a house while working.' There are chasms under foot and loose rocks overhead, equally to be avoided, and the whole shrouded in a darkness which the miner's lamp reduces only to a semi-obscurity, and which hides without removing the danger.

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*This applies more particularly to breasts worked "on batteries."—H. M. C.

"The miner's life when not at work also has its effect on his general health, and, as with every other class of men, this varies according to the tastes and temperament of the individual. His house is of frame, plainly but conveniently built, and furnished with the necessary conveniences of life. Being situated in the country, and in a section where land is of little value for either building or agricultural purposes, there is plenty of space about the house, and fresh air in abundance. Even the close neighborhood of frequent hog-pens and occasional stables, and the universal practice of emptying slops from the house on the ground at the back door, have little or no deleterious effect, being neutralized by the abundance of pure air with which their odors and gases mingle. [?]

"The miner's first care on coming from work is to take a tub-bath, cleansing his skin thoroughly. He then dresses in a clean suit, eats his supper, and is ready for the duties and amusements of the evening, both of which are few and simple. Usually the male inhabitants of the 'patch' gather in groups in the open air, in the village store, or in the omnipresent saloon, and smoke and talk, until the coming of an early bedtime sends them home. Comparatively little drinking is indulged in except on pay day, which comes once a month, and is celebrated by the drinking classes with a 'spree.' In this particular the miner's nationality makes itself seen. While men of all nations may be found drinking to intoxication, the practice as a race is confined to the Irish. There are few of American descent among the miners, and these are generally found among the best and steadiest of their class. The Irish are the most numerous, and they are fond of liquor, drink to excess, and are very quarrelsome when drunk. Terrible fights often accompany a pay-day spree among them. Next to the Irish, in numbers, are the Welsh, a temperate, thrifty, and intelligent race, who form a valuable element in the population. They are industrious and economical, generally succeed in securing homes of their own, which they delight in beautifying and keeping in order, and are apt to be found in positions of trust and authority in later life. Germans and Poles, too, are industrious and economical, but less intelligent and less temperate than the Welsh, more careless in their personal habits, and utterly regardless of the laws of health. They eat unwholesome food, sleep in ill-ventilated rooms, and early acquire a sallow, unhealthy appearance. Nevertheless, their active occupation and the enforced cleanliness of the 'shifting-suit' counteract many of the ill effects of their mode of living, and they will probably be found to average as long lives as the other races. Less numerous, though making up the bulk of the population in certain localities, are Scotch, English, and Italian miners. The last are much like the Irish in habits, while the others hold an intermediate place between them and the Welsh. It is of course to be understood that these remarks apply in general to the nationalities;* there are very good workmen and excellent citizens in all classes, and, similarly, there are worthless characters in all; but the general tendency will be found as has been stated. As in every other occupation, personal habits have their effect on the constitution, and predispose it to invite or to repel disease.

"One of the most prominent conditions of a miner's working life—certainly the first to be noticed by the casual visitor—is the absence of sunlight, a very deleterious condition as many physicians and engineers of large practical experience consider it, while others as positively deny that it has any injurious effect. Dr. J. T. Carpenter, of Pottsville, in a paper read before the Schuylkill County Medical Society, says (*Transactions Medical Society of Pennsylvania*, 1868-9, p. 488): 'The deprivation of sunlight must be a very strongly predisposing cause of disease. It is to be expected that the results of this deprivation will become apparent in general anæmia, in chronic nervous irritations, in tendencies (easily to be developed by exciting causes) toward scrofula, tubercular phthisis, and allied maladies.' Other practitioners, however, assert that the deprivation of sunlight is among the least of the miner's afflictions; that no injurious effects from it are perceptible, and that no acute disease can be traced either wholly or in part to this cause; while physicians will probably continue to differ forever as to whether or not absence from sunlight during all the working hours predisposes to or prolongs any chronic complaint. In this connection it must be borne in mind that the miner's work is carried on wholly by artificial light, and that usually of a very poor quality.

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*The Hungarian element has lately been added by the arrival of large numbers who may now be found scattered throughout the region. They are commonly ignorant, uncleanly, and undesirable,—but are generally economical.—H. M. C.

"In the winter season, especially when the mines are working full time, their inmates, as a rule, see but little of the sun during their working days. They do not complain of want of sunshine, and it is difficult to trace any ill effects of its absence upon them. Their complexions are pale, but not more so than those of persons who work at night, or in shaded rooms above ground; and their eyesight, as a general thing, considering the miserable light they have to work by, is remarkably good.

"Another evil too commonly met with in coal-mines is the cloud of dust with which the air is loaded. Where the coal is kept damp by the percolation of water, little dust is made, and the miner is comparatively free from its injurious effect; but it is exceptional for the coal to be in this condition, and it has been found that the deeper the workings penetrate the less water is found and the drier and more dusty the coal becomes. Any one who has seen a load of coal shot from a cart, or has watched the thick clouds of dust which sometimes envelop the huge coal-breakers of the anthracite region so completely as almost to hide them from sight, can form an idea of the injurious effect upon the health of constant working in such an atmosphere. Ventilation mitigates this evil, but does not obviate it, as a stream of pure water flowing into a muddy pool, of which the bottom is continually being stirred up, will thin the contents of the pool, but will not make them clear. Every fresh stroke of the pick or the hammer, every shovelful of coal moved, every fall of a dislodged mass, causes a fresh cloud of dust."

The miner works under one of two conditions; in a dry and therefore dusty mine, or in (wet or at least in damp) workings.

The train of ills that may be induced by prolonged or even temporary exposure to dampness, by wet feet and wet clothing, by cooling off too quickly when heated, by waiting in winter in the cold draught of the down-cast (often in wet clothes) for the cage to raise them to the surface, are only too well known.

The same hardships are also endured by men employed in many other occupations. Exposure to damp air, draughts, and wet clothes do not seem to have any marked effect upon a healthy man with a strong constitution,—provided he does not allow himself to become chilled, and is careful to change his wet clothing for a dry suit as soon as he reaches home.

A careful miner never loiters on the way home when his clothes or feet are wet; he goes from his work to his home without any unnecessary delay and usually at a smart gait.

While at work in the mine he rarely rests long enough to become chilled. At some collieries, some of the miners make a partial change of their clothing in the room or building provided for their use,* but this change is generally confined to some of the outer clothing (water-proof coats, boots, etc.) Wet road-beds cannot always be avoided, but there is little excuse for the overflowed gangways common in so many mines. Flooding of the gangway at certain points is caused either by obstructions in the ditch or by driving the gangway with insufficient fall to carry off the water. This is a matter that should receive the more careful attention of the mine inspectors. There is no reason why every mine should not be put in such condition that a miner with good stout boots could walk in to his working place without wetting his feet.

But while the water underfoot can be drained off, it will never be possible to stop the showers of water that pour down from the roof. To protect himself from dripping water the miner should always wear a water-proof coat;

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*See Mines Ventilation Act.

but when water drips from the roof of the breast or chamber in which he is working, he inevitably gets wet, and often very wet. In shaft sinking, and slope sinking, and other work in which the miner is working at rock or coal *beneath* him, a water-proof coat can readily be worn, but the labor required in mining from a face in front of or above the miner, generally renders such a protection at once useless and unbearably cumbersome.

Acute and chronic pneumonia, rheumatism, the speedy development of any consumptive tendency, and acute and chronic bronchitis, are the diseases most frequently developed by exposure in wet mines,—but these diseases do not seem to be of more frequent occurrence among our miners than among other classes of out-door laborers.

The laborious nature of many of the duties of the miner has often been considered as a cause (or an auxiliary) of certain diseases. The mere lifting of heavy rocks and timbers, shovelling of coal, dragging the timbers up into the workings, etc., cannot be considered detrimental to health. In the presence of some diseases any hard labor is injurious, but to a healthy man hard labor can seldom be considered harmful. It is hardly necessary to refer here to hernia, spinal injuries, etc., caused by undue exertions, as these may be caused by violent straining at any work.

The effect of breathing air loaded with carbonic acid gas has already been referred to in a preceding chapter. The ventilation of nearly all anthracite collieries is amply sufficient to dilute the carbonic acid gas to an unobjectionable amount, but in non-fiery mines the miners frequently neglect to carry the ventilating current well into the working face, and the air then becomes loaded with carbonic acid gas from the lamps and from exhalations from the lungs of the miners, and after a shot has been fired, with other noxious gases present in gunpowder smoke. In addition to carbonic acid from these sources, the coal itself doubtless gives off appreciable quantities of this gas.

There are probably very few mines in the anthracite region where the amount of carbonic acid gas present in the workings or traveling-ways is sufficient to produce any perceptibly injurious effect upon the miners.*

The temperature of workings situated at a considerable distance from the intake varies but little throughout the year. In winter the workings at the face are comparatively warm, in summer, cool, and this temperature, (representing as it does the mean annual temperature in this region) is in every way favorable to health.

Our mines are not yet sufficiently deep to render the increase of heat due to depth an appreciable factor. Even when our collieries reach a depth of 2000 or 2500 feet their temperature will probably not exceed 85° F.

Workings ventilated by a current coming by a short route from the intake, are quite cold in winter and sometimes uncomfortably warm in summer,—but the severity of the cold of winter, and the intensity of the heat of summer, are not felt in the mine as they are on the surface. In reopening mines that have been on fire, and in places where a large amount of steam is used underground, the heat in summer is very oppressive, but such conditions are exceptional.

Miners working in dry dusty workings, and breaker hands working in breakers where the coal is prepared dry, undoubtedly become the victims, sooner or later, of the disease known as "miners' consumption," "miners' asthma," "miners' anæmia," and by various other names,—a disease doubtless similar in many respects to that known as "grinder's asthma or phthisis," or "millstone makers' phthisis."

It has been noticed by many observers, from the latter part of the last century to the present time, but few have attempted to so differentiate its symptoms that it might be distinguished from other and mayhap co-existing pulmonary affections. The disease has been designated by some as "Miners' Phthisis," which seems to be a misnomer, and by such terms as "Black Phthisis," "Spurious Melanosis," or "Melanosis of Miners," "Phthisis avec Mélanose,"

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*This does not apply to abandoned mines, nor to mines communicating with workings on fire, nor to old workings in mines now working.

(Phthisis with Melanosis,) "Anthracosis,"—and diseases described under the name "Rag-pickers' or Wool-carders' asthma, Garbage-gatherers' phthisis," are evidently of similar origin.

The disease has been variously described as a chronic bronchitis due to irritation; as asthma, induced by peripheral irritation; as phthisis, and as a distinct type of fibroid phthisis or chronic pneumonia (Bristow.)

There is little doubt but that these terms have all been applied to cases of one and the same disease, having as a common origin the presence of large quantities of dust in the lungs,—with, perhaps, as a single exception, the spurious asthma originating in peripheral excitation of the nerves by small filiments of cotton or woolen fiber.

I shall endeavor to show that the disease is in reality a distinct variety of Fibroid Phthisis, known also by the names Fibroid Consumption and Chronic Pneumonia, and that it is a true cirrhosis or hardening of the lung; that it is in no respect an asthmatic affection; that it is melanic in certain cases only, and in these the melanic character is spurious *i. e.*, accidental; and that it is not bronchial, although bronchitis is often a concomitant complication.

The symptoms developed in those subjected to the inhalation of air loaded with large quantities of dust are naturally in divisions dependent upon the character of the matter inhaled; thus this material may be

1. Sharp, granular, and irritating.
2. Smooth, non-irritating material.
3. Fibrous.

Fibrous materials may possibly produce a very different train of symptoms from those caused by grains of dust,—these symptoms need not be considered here.

The inhalation of dust composed of minute grains of mineral matter must necessarily produce symptoms, the severity of which is governed by the character of the dust,—this has been shown by Hirt, (quoted by Buck,) although contra-stated by other writers,—who found that among needle polishers, file cutters, and flint workers more than *sixty per cent.* of those sick were suffering from lung disease; among grinders, grindstone makers, and stone cutters this percentage ranged from thirty to forty, while among masons and coal miners it was much less.

It seems certain that a material like quartz or corundum which pulverizes in sharp angular grains must always produce symptoms of greater severity than coal or slate dust.

This is shown by the well-demonstrated fact that pulmonary disease runs a much more rapid course among axe and knife grinders than among (school) slate workers.

The experiments of Fournié, Knauff, and Rosenthal have shown that dirt inhaled in large quantities is partially deposited in the nose and pharynx, but that the remainder passes down into the bronchi and much finds its way into the pulmonary (lung) tissue.

On entering an anthracite coal breaker where the dust is so dense that lamps were used at mid-day, the deposition of dust is mainly confined to the nose for the first two or three minutes, then, as a sense of dryness is perceived in the nostrils, the deposit extends downwards and the pharynx becomes streaked and the larynx somewhat irritated. The amount of dust retained in the upper passages probably depends upon the amount of mucous secretion. In those suffering from a copious catarrhal discharge nearly all of the dust would probably be caught in these upper passages.

When the quantity passing to the bronchi and bronchioles is small, it may possibly be removed by the ciliary action of the mucous membrane, and ejected in the sputa, but when the quantity of dust is large enough to overload this action the surplus must accumulate in the lungs. Traube suggests that the bronchial disorders induced by irritation may derange the ciliary action and incapacitate it from performing its proper work.

The experiments of Knauff, Greenhow, and Rosenthal show that matter in a finely divided state readily finds its way into the air-cells, into the ciliated epithelium, the lymphatics, bronchial glands and may even pass to the pleura. The results of these experiments are confirmed by post mortem examinations made by many observers, notably by those of Peacock and Greenhow in which exhaustive chemical tests were made to determine the identity of the interstitial deposits with the matter inhaled.

Lung tissue infiltrated with foreign substances is variously described as brownish, black, or grayish, according to the character of the inhaled matter, and is sometimes additionally darkened by the apparent presence of a species of local melanic degeneration. It is traversed by bands of nearly white fibroid material which may obliterate all signs of true lung tissue; its specific gravity is increased (Zenker) and it is much harder and more dense than normal lung tissue (Greenhow.) Hard, apparently encysted, nodules from the size of a bird-shot to that of a pea, and composed largely of the inhaled substance, are not uncommon in well-developed cases.

These lumps or nodules may originate

1. From an interstitial segregating action; or 2. By a pulmonary cell or lobule becoming thoroughly filled, occluded, and transformed into a cyst; or 3. By an ulcerative action within such a cell or lobule allowing its contents to pass bodily into the lung tissue.

The changes produced in the lung by prolonged inhalation of dust may be summarized thus:

1. A gradual infiltration of foreign matter into the lung tissue and the formation of interstitial fibroid deposits from the mechanical irritation thus produced, sometimes augmented by severe inflammation of the bronchioles.

2. Increased infiltration and condensation with concomitant diminution of the ærating surfaces. Some portions of the inhaled matter may become encysted (?) or segregated and form hardened nodules.

3. When nodules are present they may cause suppuration; the resulting abscesses then discharge into the bronchi or bronchioles followed by contraction and condensation of the pulmonary tissue with augmented fibrinous deposition. The abscesses may discharge in other directions occasioning serious complications.

That this disease is not asthma is evident, and that it simulates asthma only in its dyspnoic symptoms is equally true. The dyspnœa seems to be directly dependent upon the reduced area of ærating surface and the diminished respiratory capacity.

The mucous membrane of the air passages is exceedingly tolerant of ordinary non-fibrous dust and only in a few cases does the inhalation of such material produce severe bronchial symptoms.

When such symptoms are induced, the presence of fibroid degeneration is often not recognized, and this is probably the reason why many writers attribute the symptoms wholly to the bronchitis of irritation.

Again, others while recognizing the fibrinous deposits, consider them wholly secondary to the existing bronchial affection, and believe the fibrine is deposited around the irritated bronchioles and air cells,—but it is not uncommon to find this disease far advanced with no commensurate bronchial lesions, and as it has been shown that inhaled material easily passes to the lung tissue it seems more rational,—while not denying a possible bronchial origin,—to attribute the disease to this cause.

Symptoms.

In its early stages this disease is marked by no prominent symptoms. It is so essentially chronic, its progress is so slow (except in axe-grinders, etc.,) and the physical signs so slight, that the miner may not be conscious of any ailment until the lungs are largely infiltrated and the respiratory capacity greatly decreased by cirrhotic changes.

A slight dyspnœa, noticeable only after violent muscular exertion usually accompanied or preceded by a cough, with some chronic bronchitis and more or less expectoration colored with the material inhaled, are the first symptoms. The pulse and temperature are normal and there are no signs of emaciation or general debility, the appetite is not affected and the patient appears to be in good health ; but on examination it is found that the percussion note lacks resonance, and there is a general increased sense of resistance, while by auscultation ràles may sometimes be detected, and the respiration is decidedly enfeebled with prolonged expiration.

As the disease progresses, the dyspnœa becomes more marked, the cough may become troublesome,—but this is rare,—and hemorrhage may occur. Among grinders hemorrhage is common at this stage of the disease, but among coal miners and slate workers it is rather uncommon.

As the disease advances, all the symptoms increase in severity ; the dyspnœa is now troublesome after the slightest exertion, (mounting a flight of steps, etc.,) the temperature may be subject to temporary elevation, the appetite may fail, and vomiting occasionally occur. Hypertrophy of the right heart, with its concomitant complications, may occur if the patient is engaged at hard labor of any kind, otherwise it is rare, except in the final stage of the disease.

Severe pains from pleurisy or abscess are not uncommon. When an abscess breaks, expectoration of muco-purulent sputa containing lumps of the foreign matter may be expected.

These lumps are sometimes eliminated from the lung tissue many years after the dusty occupation has been abandoned. Coal miners freqently cough up such lumps many years after they have abandoned the occupation, and it is to this feature we owe the favorable prognosis it is almost always safe to make.

If the occupation be discarded for work in a pure atmosphere before the disease has progressed to the final stage, the symptoms rapidly disappear, and recovery may be almost entire.

If work be continued, all the symptoms continue with increased severity ; hemorrhage may frequently occur, but the amount of blood lost is usually small ; abscesses form by the breaking down of those tissues in which the pathological changes have been greatest, and these may discharge into the bronchi and bronchioles, followed by contraction and cicatrization, with propable concomitant retraction of the chest walls.

Dyspnœa is now extreme upon the slightest exertion ; the temperature may be elevated either temporarily or constantly; debility and emaciation are present in greater or lesser degree; but in some cases, while the emaciation is marked, the patient's strength continues until the very last phases of the disease, and he is obliged to relinquish his occupation but a few days or weeks before the disease terminates fatally.

While hypertrophy of the right heart may develop in the second stage of the disease, there is seldom any decided tendency to œdema or anasarca.

The formation of vomicæ is probably not unusual, but they seem to be generally much smaller than those of tubercular disease, and can have no elective tendency to form at the apices.

In uncomplicated cases death occurs from debility and innutrition; but few terminate in this way, as grave complications are almost sure to develop before the disease has time to run its course.

When the tubercular diathesis is present, there can be little doubt but that the inhalation of dust may often induce tubercular deposits. In such cases the diagnosis is most difficult, and we have a train of symptoms—emaciation, debility, fever, etc., with general dullness on percussion and marked dyspnœa,—that deceptively point to the existence of miliary disease; but their essentially chronic type will generally be a safeguard against this error.

The disease is most common (among coal miners) in persons over forty years of age. It is never well developed in children who are exempt from other lung troubles, as its march is so slow that a child will grow to manhood while the disease is in its infancy.

Among axe and knife grinders, etc., its progress is more rapid—in from three to five years it may terminate fatally.

Among slate workers it also occurs late in life. In eight cases that I have investigated, death occurred at from 42 to 61 years, averaging 53 years.

Among coal miners it may probably exist from ten to twenty-five or thirty years before reaching a fatal termination.

It can be distinguished from chronic pneumonia (fibroid phthisis) only by the history of the patient and by the expectoration of nodules of the foreign material.

From true tubercular phthisis or consumption it is readily distinguished by the *general* dullness on percussion, by the absence of fever, emaciation, etc., and by its chronic character: from chronic bronchitis by the physical signs, etc.

The great importance of distinguishing the true nature of the disease is at once apparent when we consider the favorable prognosis that can generally be given if the patient will change his occupation.

It is true that many cases terminate fatally in a short time after a physician is called to attend the patient, but this is generally owing to the fact that the disease had been making stealthy progress for many years before the sufferer became aware that his lungs were affected, and not because the disease is necessarily fatal.

No practical means of laying the dust in our mines has yet been devised, but a great improvement has lately been effected in some breakers by ventilating with a small fan.

This plan should be adopted at all breakers at which the coal is prepared dry, not only for sanitary purposes, but to enable the boys to properly clean the coal. A sufficient supply of light cannot be obtained when an impenetrable cloud of dust pervades the breaker, even if the structure is supplied with hundreds of windows.

Conclusions.

From what has been stated in the preceding pages the following conclusions may be summarized:

1. That the sanitary condition of a majority of anthracite mines is excellent.

2. That the very wet condition of some mines is prejudicial to health, but that it cannot be bettered.

3. That at many mines the condition of the road-beds might be greatly improved by draining-off standing water.

4. That the large amount of dust in many breakers is incompatible with the health of those employed therein, and that this evil might be mitigated, if not entirely removed, by ventilating these structures with small fans,—but is this advisable in very cold weather?

Treatment of Injured Miners.

Every colliery should be provided with a comfortable room above ground, and also one beneath the surface, near the foot of the shaft or slope, where injured miners or laborers can be temporarily placed for treatment or prepared for removal to their homes.

A stretcher with pillow ; a large quantity of bandages in rolls ; a lot of cotton ; lime-water and sweet oil or lime-water and linseed oil ; alum, tannin, vinegar, or some other astringent and a quantity of Monsells' salt ; sponges and clean water ; ammonia, and some whiskey, or what is much better, good brandy, are the supplies that should be kept in readiness.

In the case of fractured limbs, the injured member should be placed in a natural position, and one comfortable to the sufferer, and carried home on a stretcher. This should also be done in cases of dislocation, but the person may often best wait at the colliery until the arrival of the surgeon.

In cases of severe bruises or contused wounds, and internal injuries, the patient may be carried home, great care being taken in moving him ; but if his home is distant from the colliery, or if a surgeon can be quickly called, it is best to wait until he arrives.

Wounds on the head and other superficial wounds should be sponged out and thoroughly washed with cold water ; if there is considerable hemorrhage hot water may be tried ; if this fails use a mild astringent, but only in the most urgent case resort to the use of Monsells' salt.

When the bleeding comes from an artery the blood will spurt out in jets, and in this case an endeavor should be made to control the hemorrhage by pressure on the artery with the fingers.

When the hemorrhage is from the leg or arm a ligature should be applied. This may be readily accomplished by tying a handkerchief or muslin bandage loosely around the limb above the wound and twisting it tight with a stick.

When the hemorrhage is from a vein the blood is of a *dark color* and does not spurt out in jets. In this case the ligature is applied *below* the wound, if it cannot be controlled by pressure with the fingers.

If the patient faints from loss of blood, and the hemorrhage has greatly decreased or stopped, place him flat on his back, *do not give him any stimulant,* (brandy or whiskey,) and do not attempt to bring him to by dashing water on his face, etc. Let him lie perfectly still until the surgeon arrives. If the hemorrhage has been completely stopped by a ligature, efforts may then be made to bring the patient to, by the use of cold water, ammonia, etc., and a small dose of whiskey, (one to two ounces,) or an ounce of brandy may be given. *Place the head low.*

If there is no hemorrhage and the man is weak and nervous from the severe mental and physical shock which he has sustained, one or two ounces of brandy may be given to enable him to bear the journey home without fainting.

The sweet oil and lime-water or linseed oil and lime-water are to be liberally and promptly applied on cotton or lint to all burns, and a little brandy may be given with advantage to enable the sufferer to rally from the shock.

When a miner is found unconscious or apparently dead from the effects of black-damp or any other gas he may have inhaled, artificial respiration should be *immediately* begun and continued for fifteen or twenty minutes at least. Ammonia should be held near the nostrils and brandy given in a liberal dose. Cold water should be thrown on the face, or the face and chest may be slapped vigorously with a wet towel.

In all cases of serious injury the patient should not be moved to his home until the physician has arrived. When the injury is not serious, or is of a nature easily understood by the patient, he may be taken home as soon as he feels able to be moved.

A seriously injured person should never be surrendered to his relatives until the arrival of the physician, as they are less likely to treat him properly than any one else. They are excited and worried, and not capable of acting with discretion.

Old experienced miners, who have seen dozens of accidents and who have acquired more or less experience in handling their injured fellow-workmen, should always be chosen in preference to others, to take charge of the injured and to keep away officious and curious strangers until the physician arrives.

CHAPTER XXVI.

Preparation of coal for market.

The demands of consumers for an acceptable fuel for domestic and manufacturing purposes can only be satisfied by a more or less complicated method of preparation which has for its principal objects :

1. The removal of slate, bony coal, rock and other impurities which are present in the coal as it comes from the mine.

2. The assortment of the coal into grades of nearly uniform size.

3. As there is a larger demand for coal of the intermediate sizes than can be supplied from the coal as mined, it is necessary to break up some (or all) of the large lumps to increase the percentage of these intermediate sizes.

All of these objects are accomplished by the process of preparation in the "Breaker." By reference to the next chapter it will be seen that the breaker is a large structure in which the coal is screened over bars, picked, crushed, run through cylindrical screens, and picking shutes, and often washed, before it is finally passed to the shipping pockets.

As it comes from the mine the coal consists of fragments of all sizes, mixed with more or less slate and rock and a considerable amount of coal too fine to be marketed as fuel, and more or less coal known as bony coal, (which contains nevertheless a large percentage of carbon) and lumps of coal with layers of slate adhering to one or both sides, or distributed throughout the lump.

When the mines are very wet, as in the Panther Creek district, the appearance of a car load of coal as it comes from the mine is suggestive of anything but merchantable fuel. It seems to consist of rock, slate, and slaty coal, very little coal being visible, and the whole covered with a black muddy mass of fine coal and dirt, and dripping with dirty water.

When the colliery from which the coal is mined is opened on a thick bed with a steep dip, all the slate and other refuse is loaded with the coal, but when the bed is thin or the dip is flat, the slate, rock, and much fine stuff are left in the mine, and the coal, as it comes out of the mine, contains only a comparatively small amount of refuse

To attain the best results at the least cost, and with the least waste, it is evident:

1. That the coal already broken down to a marketable size,—say from egg to pea,—should be separated as quickly as possible from the remainder, and
2. That no coal in large lumps containing any layers of slate should be permitted to pass through the rolls
3. That all bars, screens, and rolls (coal crushers) should be fed regularly and evenly, and not overcrowded.
4. That no coal, rock, or waste should be handled or treated twice over (unless absolutely necessary).

When a mine car comes from the mine it goes directly to the dump, (cradle-dump, tip or tipple,) and the coal is dumped upon a set of inclined bars or into a shute or pocket, from which it is slowly fed under a gate and allowed to slide down over the bars. These bars are placed from three to six inches apart and separate the coal into two portions that are to be separately treated.

That portion passing through the bars is usually conveyed directly to a large but short screen, known as a dirt or mud screen, which separates the fine coal, (pea and smaller sizes.) but it is sometimes hand-picked before passing to this screen.

The broken and egg sizes coming from the mud screen are picked free from slate and sent direct to the pockets for shipment, or sent to the "Pony rolls" or "Monkey rolls" to be broken down into egg, stove, small stove, and chestnut sizes.

That portion passing over the main screen bars, runs out upon the "platform," which is a flat or slightly inclined floor covered with iron plates, and is cleaned by the platform men.

The slate and rock are picked out and sent down the rock shute, the good lump coal goes to the lump shute, and the "rough coal" (and sometimes a large part of all the lump coal) to the crusher rolls.

The method of preparation will be more fully understood by reference to the diagrams of Atlas Sheet No. XIX. These diagrams show four methods of preparing coal, which differ from each other in many particulars. These differences are mainly due to differences in the character of the coal, the amount of dirt, the wet or dry condition of the coal, etc.

A dozen other diagrams might be prepared of other breakers, no two of which would be exactly similar, but those given will serve to illustrate sufficiently well several prominent and important features. Thus we can readily see the radical difference between the method of preparing the *dry clean* coal of the Hollenback colliery and the *dirty and wet* coal at the Lehigh Coal and Navigation Company's mines, and at the same time understand why the systems vary so widely; and between the old style Hammond breaker, where the broken and egg are run completely through the breaker, and other systems where the broken (at least) is taken out at a much earlier stage,—and between the plans by which the coal is divided and subdivided and then again mixed together, and the method by which (Cross Creek) after being divided the two parts are never re-united.

Cross Creek Breaker No. 2.

The method of preparing the coal at the Cross Creek Breaker No. 2, (July, 1882,) is shown by the diagram on Atlas sheet No. XIX. At Breaker No. 1 the process is somewhat different, the principal differences being indicated by data inclosed in parentheses on the diagram.

The coal is dumped into a shute from which it is gradually allowed to run down upon the bars under a gate provided with a row of sharp (spike) teeth on its lower edge. This gate has a vertical motion, moving between upright guides, and is operated by a hand lever.

The bars are placed from three to four inches apart, and divide the coal into two portions.

That portion passing over these bars passes on to a set of bars placed about seven inches apart,—these bars forming a continuation of the first sets;—the coal passing over these bars is necessarily of large size, and runs directly out upon the platform. Here the slate and rock are separated by the platform men and the coal is pushed down into the lump coal shute. At the head of this shute another set of seven-inch bars is placed; the coal passing over these bars runs down into the lump coal shute to be shipped as lump, while the smaller pieces run through these bars and pass into the steamboat rolls, which crush it down to the size of steamboat and smaller. From these rolls it passes to a shute in which are bars nearly five inches apart which separate out the smaller sizes (but before reaching these bars the slate is picked out) and allows the steamboat to pass over and down into the steamboat shute. The smaller sizes passing through these bars runs into a pair of small rolls, is crushed down to the size of broken and smaller, and passes into the broken screen.

The coal that passes through the first set of seven-inch bars runs into a shute and passes to a second set of bars nearly five inches apart, the slate being picked out on its way down the shute,—the steamboat coal passes over these bars to the steamboat shute, and coal of smaller size passes through to a pair of small rolls breaking it to the size of broken and smaller, and then passes to the broken screen.

The broken screen obtaining coal from the above two sources, separates out broken and egg sizes and all the smaller sizes pass direct to the stove screen.

The broken and egg both pass down picking shutes ("*telegraphs*") to their appropriate pockets.

The stove screen separates out stove and chestnut sizes, and the smaller sizes pass to the pea screen.

The stove and chestnut both pass along picking shutes on their journey to their appropriate pockets.

The pea screen separates out pea and buckwheat and dirt.

The pea and buckwheat sizes both pass to Clark jigs and from the jigs to the loading pockets, and the dirt (passed through $\frac{3}{16}''$ mesh) goes direct to the dirt pocket and thence to the dump.

The coal that passes through the first set of bars is very dirty. It goes direct to the mud screen which separates,

1. Coal larger than broken ; (over $3\frac{3}{4}''$ mesh ;) this goes to a set of small rolls and thence to the broken screen.
2. Broken and egg sizes ($3\frac{3}{4}''$ to $2''$) which goes direct to the broken screen.
3. Stove, and smaller sizes going to the stove screen.
4. Pea and smaller.

The stove screen separates stove and chestnut which are both jigged before going to the pockets, and pea and smaller sizes which with the same sizes separated out by the mud screen (4) go to the pea screen.

This pea screen separates in the same manner as the one already described.

The broken screen also separates in the same way and the coal is put through the same process as that already described, the smaller sizes passing successively to stove and pea screens.

Hammond Breaker.

The method of preparing the coal at Hammond colliery given below is condensed from a description by Mr. H. S. Thompson, Mining Engineer for the Girard Estate :

It will be easily understood by reference to Atlas sheet No. XVIII, and to the diagram on Atlas sheet, No. XIX. The letters all refer to the plan and elevation of this breaker shown by the former illustration.

At the Hammond colliery the product of the mine is raised through the slope in small cars, called wagons, which run by gravity from the head of the slope to the breaker tips or dumps, two in number, A, A, where they are dumped and emptied in the manner shown. The empty wagons are pushed back by hand to the foot of the automatic plane B, by which they are raised to a height sufficient to allow them to return by gravity to the head of the slope.

The mixed coal, slate, and dirt taken from the mine enters the "dump shute," C, C, and reaches first the "main dump shute bars," C^1, which are narrow cast-iron bars, so set as to allow a space of two and a half inches between each two of them. Most of the material small enough to do so, passes through these bars to the dump shute "hopper" D, from which it is fed into the counter screens, E, E, one on each side, with their supplementary screens, E^1, E^1, and E^2, E^2.

In the counter screens, E, E, the first segments extract all smaller than and including the chestnut coal. The pea coal and chestnut coal are separated from the dirt in the supplementary screens, E^1, E^1. The other segments separate the large and small stove from the egg coal and larger pieces, which pass out at the ends of the screens, while the stove coals are again cleaned of smaller particles and flat slates in the supplementary screens, E^2, E^2.

The dirt from all these screens passes down the counter dirt shute, D^5, and is hauled away on the tramway, Z, across the trestling, Y, Y, to the dirt bank, in small cars, which may be tipped on either side, called "dumpers."

The slate and bone separated from this coal in part by the so-called slate picker screens, and in part by hand picking, containing a considerable quantity of good coal mixed with or adhering to it, is collected in the slate picker hopper, D, from which it is taken along the tramway, Z^1, to the boiler fires, and used as fuel.

When it leaves the counter screens the coal descends, by its own weight, along narrow troughs, FF, called "telegraphs," the hand picking being done by men and boys at convenient places along them, and is distributed thus:—the pea and chestnut to the main pea and chestnut coal screeens, L^1, L^1, L^1, the large and small stove to the main screens, L, L, and the egg and broken to the prepared coal rollers, K^1.

As the counter screen material is usually wet, and the dirt adheres to it, this second screening is necessary in order to clean it properly. This completes the distribution of that portion of the wagon contents which passes through the two and a half inch openings between the bars, C^1, C^1, which is only the smaller sizes from egg down, with such of the larger thin slabs of coal as may turn edgewise and pass through the bars.

That portion of the wagon contents which passes over the bars, C^1, C^1, goes to the steamboat bars, C^2, (on its way down to the dump shute,) which are set four and a half inches apart. All that passes through the bars, C^2, goes to the bars, C^3, set two and a half inches apart, through which all below, and including, the egg coal passes to the dirt screens E^5, where the dirt is separated from it, the coal going to the prepared coal roller, K^1, and main screens, L, L, while the dirt goes into the hopper, D^4, and thence to D^5.

By this process coal that should have passed through the bars, C^1, but has been crowded over them, is separated in the dump shute, C, C, by bars, C^2, and is removed and distributed as described. That portion which passes over the bars, C^3, goes to the principal steamboat bars, C^4, where it joins the coal from the steamboat roller, K. The remainder of the wagon contents goes down the dump shute to the platform bars, C^5, and contains nothing smaller than lump coal. The platform bars are set nine inches apart, and that portion which passes over them goes to the first platform, H, where such of the lumps as are suitable are pushed into the lump coal shute, I, and the slate and rock into the rock shutes, I^1, I^1, on either side. Here, also, such of the lumps as have streaks of slate or bone through them are broken by hand, and the impurities removed. All that is not suitable for lump coal is thrown down a hole in the platform to the second platform, H^1, where it joins that which passes through the bars, C^5. Here the slate is carefully picked out again by hand and sent down the rock shutes, I^1, I^1, to the point, I^2, where it is loaded into dumpers, and hauled to the dirt or rock bank.

The lump coal passes down the lump coal shute, I, I, to the point, I^3, where it reaches such a level as will allow it to be loaded into the railroad cars for market. After the slate has been removed at H, the coal is thrown down a hole in the platform to the steamboat rollers, K, set so near together that none of these pieces shall be larger than steamboat coal. The coal passes from the roller, K, to the principal steamboat bars, C^4, set four and a half inches apart, and all that will not go through them is steamboat coal, and goes to the steamboat shute, G, whence it is loaded into the railroad cars at G^1, in the same manner as is done with the lump coal at I^3, the same track answering for both. That coal which passes through the steamboat bars, C^4, goes to the prepared coal rollers, K^1, which produce no coal larger than broken. From these rollers the coal enters the main screens, L, L, which are similar in construction to the counter screens described above, except that they are larger. The dirt and pea and chestnut coals are taken out in the first three segments, and go together to the pea and chestnut screens, L^1, L^1. L^1, which are double screens having a surrounding mesh, outside the screen, called a "jacket." All but the chestnut coal passes through the inner screen, the chestnut dropping out at the end, and the dirt passes through the jacket, while the pea coal drops out at the end of the jacket. The dirt drops into the dirt hopper, D^2, and is hauled to the dirt banks; the pea goes to the pea coal bins, N^6, and the chestnut to the chestnut coal bins, N^5, without further preparation.

The next segment in each main screen separates the small stove, the next the large stove, the next the egg, and the broken coal falls out at the end.

All the coal but the broken passes, each size separately, over short "dusting bars," M^1, set three eighths of an inch apart, which remove the dirt made by attrition in the screens, and thence over the picking floors, M, M, which are so inclined that the coal slides over them by gravity, each in its own compartment.

These compartments are:

M^5 M^5, small stove picking compartment.
M^4 M^4, large " " "
M^3 M^3, egg " "
M^2 M^2, broken " "

Across the picking floor, seats, M^6, M^6, are placed, which are slightly elevated above them, so that the boys who pick the slate sit above the coal and separate the slate from it by hand as it passes them on its way to the bins. The slate picked out by the boys is collected in the slate shutes, M^7 M^7, and carried by hand to the dirt hopper, D^2.

The bins, N, N, are divided into compartments for each size of coal, N^1, N^1, being the broken coal bins ; N^2, N^2, the egg coal bins ; N^3, N^3, the large stove coal bins ; N^4, N^4, the small stove coal bins ; N^5, N^5, the chestnut coal bins ; N^6 N^6, the pea coal bins ; N^7, the buckwheat coal bin. When it is desired to load this coal into the railroad cars, they are run down under the breaker to the gates, O, O, communicating with each bin, the car loaders using the platform, P, for convenience in walking from bin to bin. Below each of the gates, O, is a short set of bars over which the coal passes on its way into the cars, called the "lip screens," which take from each size all particles smaller than it, as well as the dirt caused by attrition in the bins. There are also similar bars at the lump and steamboat loading points, I^3 and G^1. All this "loading chippings" and "dirt," as it is called, are brought to the hopper of the elevators, R, R, the lump coal chippings by means of the swinging trough, S ; the steamboat chippings by a trough not seen in the drawing, and the dirt from the lip screens at O, O, by wheelbarrows. It is then all elevated by the elevators, R R, and carried horizontally by the chain carriers, T, T, to the bars, V. That which passes over the bars goes to the so-called "monkey rollers," K^2, which are similar to the others described above, but smaller ; while that which passes through goes directly to the elevators, R^1, R^1. After passing through the monkey rollers, that portion goes also to the elevators, R, R, and all together are elevated to the buckwheat screen, W. All but the dirt and buckwheat coal passes through this screen and goes to the main screen, L. The buckwheat passes through the screen and drops from the jacket to its proper bin, N^7, while the dirt passes through the jacket and goes to the hopper, D^2.

None of the diagrams on Atlas Sheet No. XIX have been made to show the method of rehandling and separating the coal and dirt that collect beneath the dusting bars and lip screens.

L. C. & N. Co.'s Breaker No. 10.

The method of preparing the wet dirty coal of the Panther Creek basin is shown by the diagram on Atlas Sheet No. XIX.

From the dump the coal passes down over bars placed about five inches apart. That portion passing over these bars runs out upon the platform where the slate and rock is picked out and sent down the rock shute; the "rough coal," which is coal with a dirty dingy look, or coal that is full of cleavage planes and appears to be shattered, is separated and thrown down a hole in the platform to the large rolls, and the lump is pushed over into the lump shute, at the head of which bars three inches apart separate out the smaller sizes, which pass directly into the large rolls.

The portion that passes through the first set of (5″) bars passes directly to a second set of bars three inches apart; the material passing through these bars contains a large amount of dirt and goes to the mud screen, while the larger sizes passing over the bars, runs out on a picking table where the slate and rock is separated and then passes to the large rolls.

All the coal from the large rolls passes to the broken screen, which separates out the broken and allows all the smaller sizes to pass through. The broken runs out on a picking table and thence to its pocket; the smaller sizes pass into the elevator hopper or pit.

The mud screen simply sizes the coal into four sizes:

1. Broken, which is picked, and then passes to the monkey rolls.

2. Egg, which is jigged, and then passes to the monkey rolls.

3. Stove, also jigged, and then passes to small monkey rolls.

4. Chestnut and smaller sizes, which pass to the chestnut screen.

The chestnut screen separates out chestnut, which passes direct to the elevator hopper or pocket, and then pea and smaller sizes pass to the pea screen.

All the coal that passes through the monkey rolls and small monkey rolls also passes to the elevators.

The elevators therefore receive coal of mixed sizes from egg down to dirt. They elevate it to the main screens into which it is then directly passed.

This screen separates:

1. Egg, which is jigged, then picked, and passed to its pocket.
2. Stove, which is treated in the same way.
3. Chestnut, which is jigged and sent to the chestnut pocket.
4. Pea and smaller sizes, which pass to the pea screen.

This latter screen separates the pea and buckwheat, (which are passed directly to their appropriate pockets,) from the dirt.

Hollenback Breaker.

The coal is here dumped and run over screen bars five inches apart. Coal passing *over* these bars runs down to the platform where the slate and rock is picked out, the rough coal thrown into the crusher (prepared coal) rolls, and the lump coal sent down the lump coal shute.

Coal passing *through* the bars goes to the mud screen. This screen has a large (5″) mesh inside. Coal passing *over* this mesh goes to the prepared coal rolls; coal passing *through* this mesh but *over* $2\frac{3}{4}$-inch mesh (the outside jacket) is steamer coal, or is sent to the rolls to be broken. Coal passing through the $2\frac{3}{4}''$ mesh goes to the broken screen.

This screen also receives the coal that has passed through the rolls. It separates out egg,—which is picked and goes to the pocket,—coal passing through the egg mesh, which goes to the main screen. Coal coming out of the end of the broken screen goes to the pony (prepared coal) rolls and then passes to the main screen.

The main screen separates egg, large stove, and small stove, which are each picked in picking shutes and go to the pockets; and the smaller sizes go to the chestnut screen to be separated, going thence direct to the pockets.

Size of Screen Meshes.

At the Cross Creek Breaker the coal is sized by passing it over and through meshes as follows:

Lump,—over bars about 7″ apart.

Steamboat,—through bars about 7″ apart, and
over bars about 5″ or less apart.

Broken	over a mesh $2\frac{5}{8}''$,	and through a mesh $3\frac{1}{2}''$ square.
Egg	“ “ $2''$,	“ “ “ $2\frac{5}{8}''$ “
Stove	“ “ $1\frac{1}{8}''$,	“ “ “ $2''$ “
Chestnut	“ “ $\frac{3}{4}''$,	“ “ “ $1\frac{1}{8}''$ “
Pea	“ “ $\frac{1}{2}''$,	“ “ “ $\frac{3}{4}''$ “
Buckwheat	“ “ $\frac{3}{16}''$,	“ “ “ $\frac{1}{2}''$ “

At the Lehigh Coal and Navigation company's breakers the meshes are about as follows:

Broken	over a mesh of $2\frac{3}{4}''$,	and through ——
Egg	“ “ “ $1\frac{3}{4}''$,	“ “ $2\frac{3}{4}''$
Stove	“ “ “ $1''$,	“ “ $1\frac{3}{4}''$
Chestnut	“ “ “ $\frac{3}{4}''$,	“ “ $1''$
Pea	“ “ “ $\frac{1}{2}''$,	“ “ $\frac{3}{4}''$
Buckwheat	“ “ “ $\frac{1}{4}''$,	“ “ $\frac{1}{2}''$

At Pardee and Co.'s Hazleton No. 6 colliery the sizes are about as follows:

Broken	over a mesh of $2\frac{1}{2}''$,	through ——
Egg	“ “ “ $1\frac{7}{8}''$,	“ $2\frac{1}{2}''$
Stove	“ “ “ $1\frac{1}{4}''$,	“ $1\frac{7}{8}''$
Chestnut	“ “ “ $\frac{3}{4}''$,	“ $1\frac{1}{4}''$
Pea	“ “ “ $\frac{1}{2}''$,	“ $\frac{3}{4}''$
Buckwheat	“ “ “ $\frac{3}{8}''$,	“ $\frac{1}{2}''$

Many collieries in the Schuylkill district make the separating meshes about as follows:

Broken	over a mesh of $2\frac{3}{4}''$,	through ——
Egg	“ “ “ $2''$,	“ $2\frac{3}{4}''$
Stove (large)	“ “ “ $1\frac{1}{2}''$,	“ $2''$
Stove (small)	“ “ “ $1''$,	“ $1\frac{1}{2}''$
Chestnut	“ “ “ $\frac{5}{8}''$,	“ $1''$
Pea	“ “ “ $\frac{3}{8}''$,	“ $\frac{5}{8}''$
Buckwheat	“ “ “ $\frac{1}{4}''$,	“ $\frac{3}{8}''$

In the Wilkes-Barre district the Lehigh and Wilkes-Barre coal company have used the following:

Broken	over a mesh of $2\frac{3}{4}''$,	through ——
Egg	“ “ “ $2\frac{1}{4}''$,	“ $2\frac{3}{4}''$
Stove (large)	“ “ “ $1\frac{3}{8}''$,	“ $2\frac{1}{4}''$
Stove (small)	“ “ “ $1\frac{1}{4}''$,	“ $1\frac{3}{8}''$
Chestnut	“ “ “ $\frac{5}{8}''$,	“ $1\frac{1}{4}''$
Pea	“ “ “ $\frac{3}{8}''$ to $\frac{1}{2}''$,	“ $\frac{5}{8}''$

Making no coal of buckwheat size.

It will at once be seen that the sizes made in different districts are not uniform.

The following figures give about the maximum range in the sizes of mesh used for separating coal of different sizes for market.

Lump ; over bars placed 7 to 9 inches apart.

Steamboat ; over bars placed $3\frac{1}{2}$ to 5 inches apart and through bars seven inches apart.

Broken ;	over a mesh	$2\frac{3}{8}''$ to $2\frac{7}{8}''$,	through a mesh or bars	$3\frac{1}{4}''$ to $4\frac{1}{2}''$
Egg ;	“ “	$1\frac{3}{4}'$ to $2\frac{1}{4}''$,	“ “	$2\frac{3}{8}''$ to $2\frac{7}{8}''$
Large stove ;	“ “	$1\frac{1}{4}''$ to $1\frac{7}{8}''$,	“ “	$1\frac{3}{4}'$ to $2\frac{1}{4}''$
Small stove ;	“ “	$1''$ to $1\frac{1}{4}''$,	“ “	$1\frac{1}{4}''$ to $1\frac{1}{2}''$
Chestnut ;	“ “	$\frac{5}{8}''$ to $\frac{3}{4}''$,	“ “	$1''$ to $1\frac{1}{4}''$
Pea ;	“ “	$\frac{3}{8}''$ to $\frac{5}{8}''$,	“ “	$\frac{5}{8}''$ to $\frac{7}{8}'$
Buckwheat ;	“ “	$\frac{3}{18}''$ to $\frac{3}{8}''$,	“ “	$\frac{3}{8}''$ to $\frac{5}{8}''$
Dirt :	“ “	— —	“ “	$\frac{3}{16}''$ to $\frac{3}{8}''$

Collieries making no small stove coal, make the chestnut meshes somewhat larger than usual, and the stove coal is made up of pieces varying widely in size,—thus the stove coal in some cases contains all that will pass over a mesh of $1\frac{1}{8}''$ or $1\frac{1}{4}''$ and through a mesh $1\frac{7}{8}''$ to $2''$.

Only a small quantity of steamboat coal is marketed, and many collieries never make coal of this size. When steamboat is not made the broken coal is usually made larger than otherwise. This is accomplished not by enlarging the mesh *over* which it passes, but by placing the bars through which it passes further apart, or if it passes through a mesh, by using a larger mesh.

We may now consider some of the distinctive features shown by the diagrams of Atlas sheet No. XIX, but before proceeding to these it may perhaps be well to mention the fact that a portion of the slate is often removed by the screens, and that this is not shown by the diagrams.

A small segment of the screen is made with long narrow slits instead of a square mesh; as a large percentage of the slate fragments are *flat*, they fall through these aperatures (through which the coal cannot pass) into a slate shute.

An examination of the diagrams shows at once that all the material passing through the first set of bars, and which usually goes direct to the mud or dirt screen ("Egg Counter" at Hammond) is subjected to a much more complicated (picking and jigging) process for the separation of bone, slate, rock, etc., than the portion passing over these bars.

The latter consists of large pieces and is cleaned on the platform, (or in picking shutes) before being broken by the rolls, and the smaller sizes made by rolls should contain only a small amount of slate and other impurities; but the portion going to the mud screen contains all the fine dirt, and small fragments of slate and rock coming from the mine.

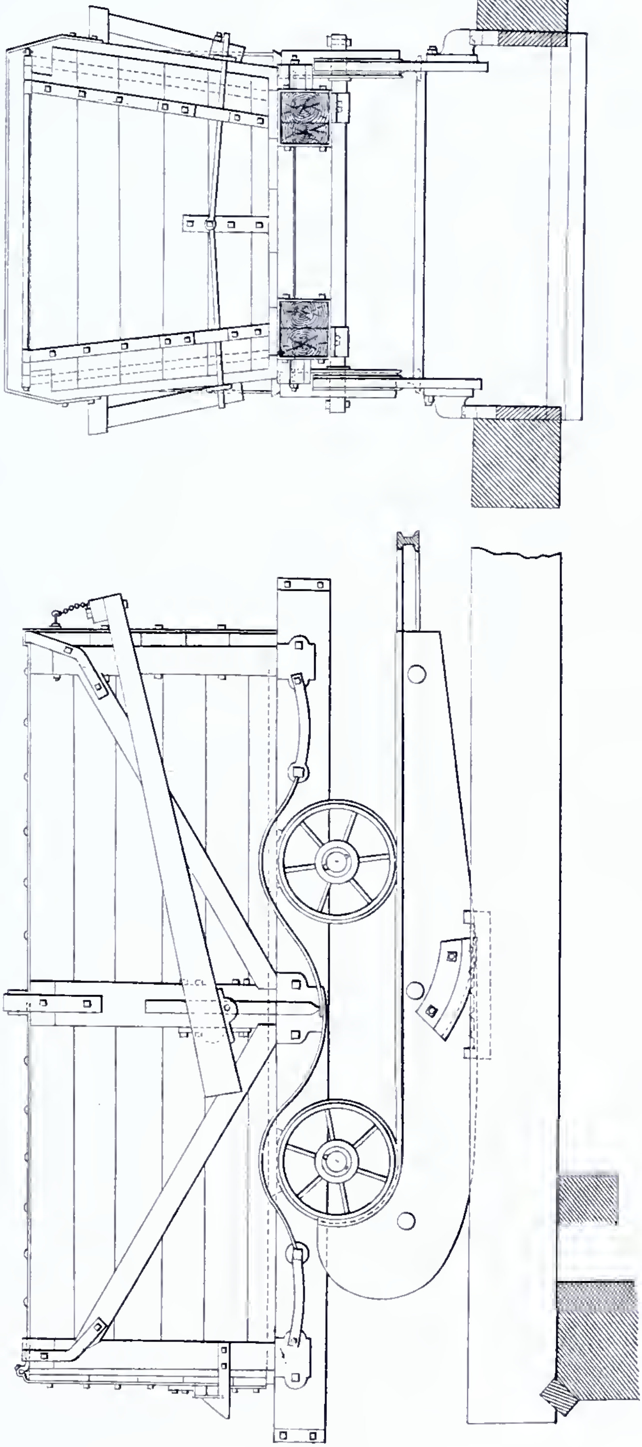
When the coal is dry it is rarely subjected to any washing process; but when wet, the fine dirt adhering to each fragment makes washing necessary, not only to remove this dirt from the coal sent to market, but to wash the coal clean so that the slate pickers can distinguish between slate, coal, and bony coal.

It will also be observed that coal is sometimes jigged and then picked to free it from slate. If the jigging was entirely successful, or even approximately so, this would be unnecessary, but the specific gravities of coal and of slate do not differ sufficiently to insure perfect separation by jigging. Rounded fragments of slate, shale, and rock are easily separated, but *flat pieces* (especially of slate) are buoyed up by the water and pass over with the coal,—hence picking is necessary to remove it. The "slate-picker," composed of narrow slits, will remove a considerable quantity of this material, but if the coal is "shelly" and breaks into flat pieces it allows a large amount of coal to escape from the screen along with the slate.

Jigging seems to be the best means of removing the slate and other rocky impurities. By this method the rounded or lump-shaped pieces can be taken out, and the flat pieces of slate coming over the lip of the jig with the coal are easily seen and removed by the slate pickers stationed along the telegraph or shute through which the coal runs on its way to the pocket.

The cost of preparing coal for market is governed by the character of the coal, the percentage of refuse it contains,

CRADLE-TIP FOR DUMPING CARS.



the percentage of large sizes (lump, steamboat, and broken) sent to market, etc., etc.

While at a few collieries coal is sometimes prepared at a cost of eight or ten cents a ton, the average cost is probably between fifteen and twenty cents. At some collieries the cost is very much in excess of these figures, but I have not been able to obtain figures showing the cost of preparing wet dirty coal for domestic use.

I have not been able to obtain the percentages of different sizes shipped from the whole region.

At some collieries all the coal is broken down and shipped for domestic use in sizes smaller than steamboat ; at others 40 or 50 *per cent.* or more of the output is shipped as lump, (for furnace use, etc.,) and eight or ten *per cent.* shipped as steamboat for manufacturing and other purposes.

Chapter XXVII.

The Anthracite Coal Breaker.

The term "coal breaker" was at first applied to the rolls or crushers (jaws) by which the coal was broken or crushed down to small sizes. These crushers or rolls were commonly located in a structure containing the screens and other machinery for separating and cleaning the coal, and in a short time these structures received the name applied at first to the crusher or rolls ; so that the term "Breaker," "Coal Breaker," or "Anthracite Breaker" is a name now given to the *structure* containing the crushing, separating, and preparing appliances, and is sometimes also understood to include these appliances,—in the same way that we commonly use the term "*mill*" (flour-mill, woolen-mill) to include both the structure and machinery,—but the term "breaker" is now never applied to the rolls by which the coal is broken. These latter are called "rolls" or "crushers."

Two very different styles of breaker are shown by Atlas plates Nos. XVI and XVIII ; breakers in process of construction by Page plates Nos. 45 and 46, and finished structures (in perspective) by Page plates Nos. 43 and 44.

The height of a breaker is necessarily governed by the method of preparation. It is always desirable to handle the coal entirely by gravity, by allowing it to slide down shutes from each set of bars, or rolls, or screen, etc., until it reaches the pockets; but when the coal is very dirty, the method of preparation is sò complex that to handle the coal in this way would require structures of inordinate height. Thus at the Lehigh Coal and Navigation Company's Breaker No. 10 one part of the coal passes through the following appliances:

Bars 5″, shute, bars 3″, shute, mud screen, shute, jigs, shute, small monkey rolls, shute, elevator, main screen, shute, jigs, shute, picking table, shute, pocket.

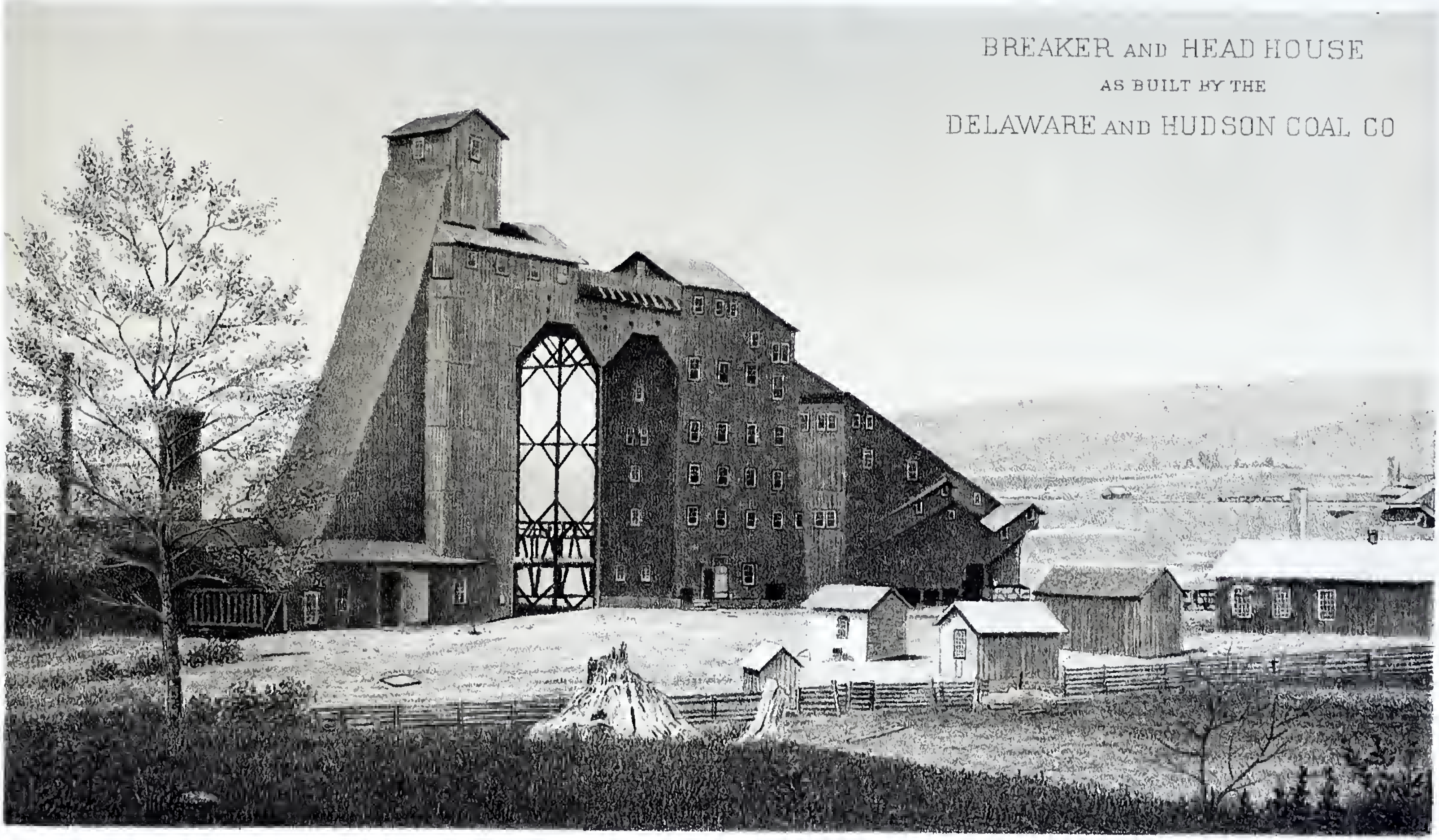
Under such circumstances an elevator is commonly used to raise the coal to the main screen, which is placed at sufficient height to allow the coal to run by gravity through the shutes, etc., to the loading pockets; but when the coal is dry, or when it comes from a colliery opened on dips less than thirty or thirty-five degrees, the method of preparation is more simple and elevators are not often necessary.

Anthracite breakers as built at present range from sixty to one hundred and fifteen feet in height; large breakers are seldom less than eighty feet high.

The necessary height is estimated from the level of the railway track on which the coal is loaded. The base of the coal pockets must be from nine to twelve feet or more above the track, so that the loading apron will project nearly over the center of the car when at an angle of twenty-five or thirty degrees, and will still be about one to two feet above the top of the car.

The pockets are made hopper (V) shaped with a slope of from twenty to thirty degrees so that they can be readily emptied. Their depth depends on the capacity of the breaker and its width, the facility for obtaining a regular supply of railway cars, etc., varying from ten to twenty feet or more.

The shutes through which the coal runs in passing through



BREAKER AND HEAD HOUSE
AS BUILT BY THE
DELAWARE AND HUDSON COAL CO

COAL BREAKER AS BUILT BY THE D & H C CO

different parts of the breaker are lined with plate or sheet (or cast) iron, and must have a pitch of from three to seven or eight inches per foot. Clean dry coal will run nicely on an iron shute with a pitch of 3 to $3\frac{3}{4}$ inches in twelve, the larger sizes running more freely than pea and buckwheat. Wet coal requires a slope of $3\frac{1}{2}$ to 5 inches in twelve and small sizes are inclination of 5 to 7 inches in twelve.

The vertical height required for free sliding is therefore not only governed by the length of the shute but by the character of the coal and its method of preparation, and also by the angles in the shutes. At corners or sharp turns the inclination of the shutes is made greater than at other points.

The length of the shutes (which are often called "telegraphs") is also governed by the character of the coal. If it contains much slate the telegraphs or "picking shutes" are made long enough to allow room for from four to six boys between the screen and the pocket. Shutes not used as picking shutes are made as short as possible by locating the screens, rolls, jigs, etc., as near each other as possible, so that one screen may feed almost directly into another screen, etc.

The screens are given an inclination of about one in twelve, ($\frac{3}{4}''$ to $1\frac{1}{2}''$ in $12''$,) and this adds a few feet to the height. The bars, platform, rolls, and feeding hoppers, all add to the height, and it is now considered advisable to extend the main dump shute for a considerable distance above the bars, to provide a pocket or reservoir for the coal from which it may be allowed to run out upon the bars slowly and regularly.

Anthracite breakers are always frame structures built of pine, hemlock, or oak* timber. Atlas Sheets XVI and XVIII show sufficiently well the method of construction. Very heavy timber is sometimes used in the lower part of the breaker to support the pockets and superstructure, but it is now considered better to replace these very heavy sticks by two smaller timbers placed skin to skin. When this plan is adopted a defective stick can readily be removed and replaced by a sound one, even when the pockets are full.

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* I believe some Southern pine is now being used.

The cost of an anthracite breaker, including machinery, ranges from twenty-five to one hundred thousand dollars. Large breakers for dry coal commonly cost from forty or fifty to eighty thousand dollars; the cost is greater for breakers for wet coal.

In capacity they range as high as two thousand tons per day, but there are only a small number actually preparing more than one thousand tons per day.

The capacity of a breaker is rarely limited by the capacity of its rolls or crushers, as these are usually capable of crushing more coal than the screens and jigs can properly prepare.

In considering the details of the coal preparing machinery, I shall begin at the top with the tip or dump, and follow the coal down through the screens, jigs, and other appliances, to the pockets.

The Method of Dumping.

Page-plate No. 42 shows the ordinary cradle-dump. It is hung on two adjustable rockers, resting on plates provided with teeth that fit into indentations on the rocker, to prevent it from slipping. As the car runs upon the dump, one of the head-men (top-men) knocks up the door latch or fastening; as soon as the coal has all run out,—and it is often necessary to shovel out a portion of the load,—the dump is pulled down by a lever, and the car run off.

A simple method of automatically opening the door latches, and at the same time, of holding the door up out of the way so that the coal can run out freely, is shown by Fig. 59.* It is used by the Lehigh Coal and Navigation company.

The door latches are made to project three or four inches beyond the sides of the car. When the car runs upon the dump the latch, *a*, is unfastened by striking against the inclined surface of *b*, up and along which it runs and passes over the knuckle and upon *c;* the car then begins to turn

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* Figure not drawn to scale—it is a mere sketch,—the letters are omitted.

up, and the latch runs down along b to the point a, holding the door up out of the way as shown by the sketch.

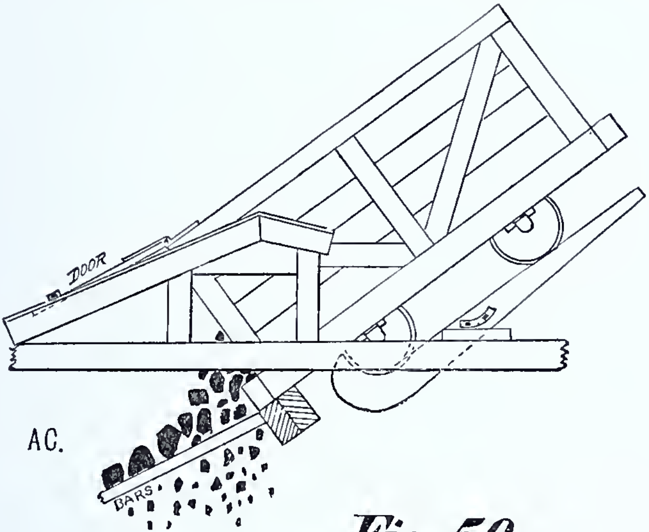


Fig. 59.

At breakers not built over the shaft or slope, the coal is frequently raised to the top in a dumping cage. Two styles of dumping cages are shown by Figs. 3 and 4 on Atlas Sheet No. XI. Their construction and operation are sufficiently well shown by the illustrations and need not be described here.

When the coal is raised in gunboats or self-dumping slope cars, or the dumping is effected by a barney, a considerable saving is effected at the top. Only one man or a man and boy are then necessary, and at some slope collieries the top men are entirely dispensed with.

When the breaker is located immediately over a shaft, or in line with a slope, four tracks are generally needed, so that the loaded cars may be run directly to the dump, a side track for empty cars being placed on each side.

At shaft collieries raising a large quantity of coal, it is of great importance to handle the cars quickly at the top, so that a minimum amount of time is spent in running the loaded car off the cage and in running the empty car on. This is best accomplished by starting the loaded car by running the empty car into it from behind, the tracks being laid with a grade sufficient to insure easy handling of the loaded cars.

The empty cars are taken around behind the loaded cars into a back-switch, or they may be shifted by a truck, as shown by Atlas Sheet No. XV. A back-switch is for many reasons preferable to a transfer truck, but requires much more space than the latter. The truck at the Hollenback breaker was added after the breaker was built. This accounts for the presence of the deflection sheave shown by Atlas Sheet No. XVI.

When a loaded car is raised, the empty car standing on the truck is started, and, running down the grade, collides with the former, starting it down towards the dump. The empty car runs upon the cage, is locked in place, and lowered into the mine while the loaded car is being dumped. As soon as the car is emptied it is hooked to the rope by a chain, as shown at *j*, the lever *a* is thrown forward or backward as the case may be, throwing *f* or *f′* into gear, thus communicating motion to the drum *c*. The car is hauled (up grade) by the rope along the side track *s* or *g* and run upon the truck.

The truck is then moved over by means of the lever *b*, the bevel gear *l*, *k*, and the rack (*m*) and pinion, until the car stands immediately behind the compartment from which it was raised.

The latches at *p*, and on the track *x* are spring switches; the latches at *g* remain in the position shown except when a car containing rock, slate, or other refuse is raised, when the car is back-switched through *g* to the rock dump.

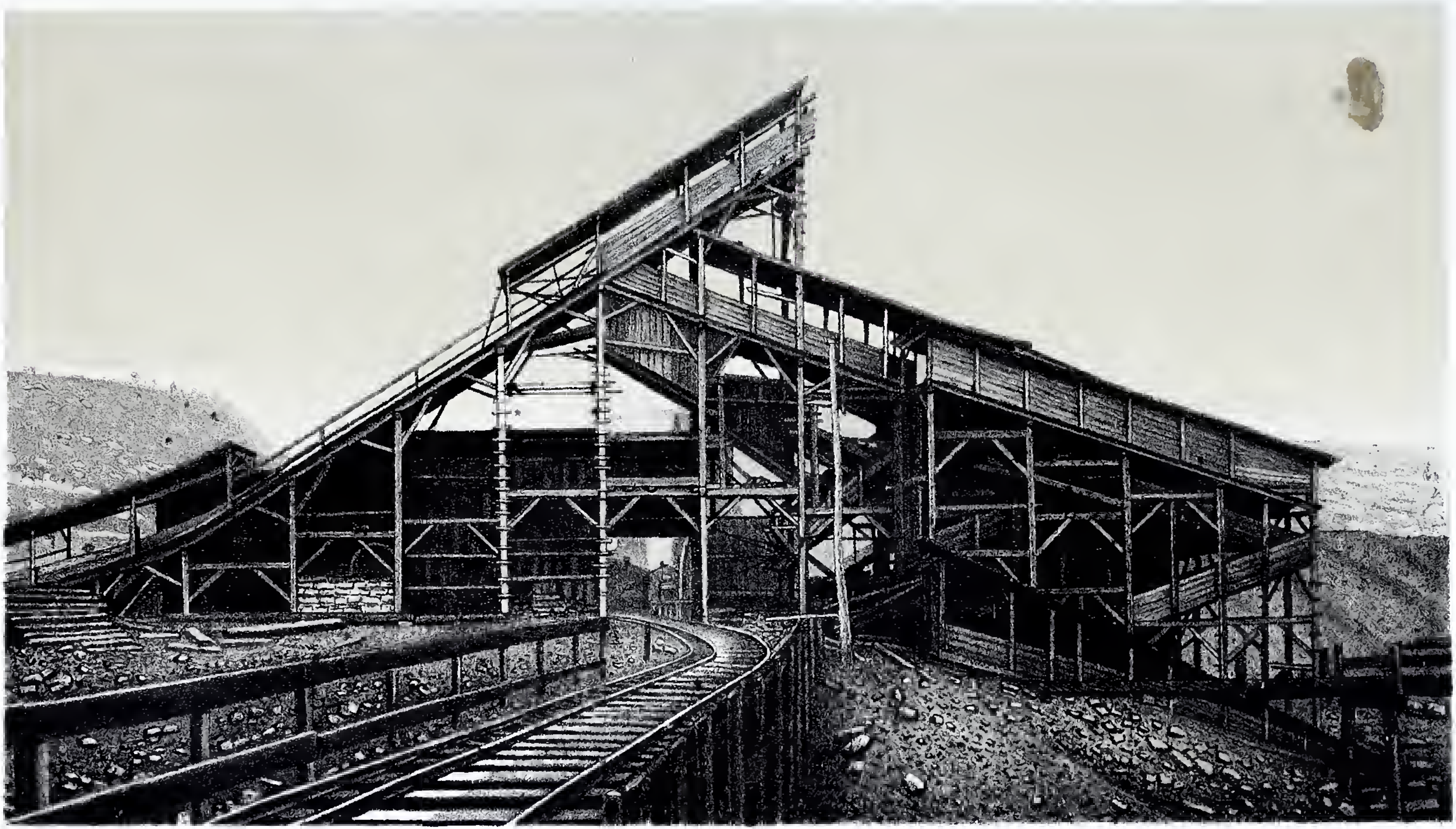
This is not the plan usually adopted for handling the refuse. Cars filled with refuse are seldom raised to the top of the breaker, but are commonly taken off at a landing level with the ground, or with the culm car tracks.

The illustration (Plate XV) shows the course of the rope around small sheaves at *d*, *e*, *i*, etc.

The levers *n* and *n′* operate the cage rests (keeps, wings) for the two compartments, as shown at *r*, *r*.

Shute linings.—The main shute into which the coal is first dumped, (when not dumped directly on the bars,) the

Second Geol. Survey of Pa. *Report A.C. Page Plate No. 45.*



lump coal and steamboat shutes, the shute leading from beneath the bars to the mud screen, and in general all shutes in which large coal is handled, require stout plate-iron linings. When the coal is wet and the water very acid, cast-iron plates are occasionally used, but the most common plan is to line with wrought-iron plates.

The main dump shute, steamboat, and lump shutes, are not infrequently lined with heavy strap-iron placed as closely as possible to cover the entire bottom of the shute, the sides being lined with plate iron.

Hoppers, shutes, and telegraphs in breakers preparing dirty wet coal that must be washed and jigged with strong mine water, are made of cast-iron three-fourths to one-and a-quarter inches thick. In the Panther Creek district plates of this thickness are often destroyed by the mine water in one year.

Breakers preparing dry coal have their hoppers, shutes, and telegraphs lined with sheet-iron plates.

Screen Bars.

The "screen bars," "bars," or "grate bars," are made of various forms.

The object of screening over bars is to separate, at once, coal already broken down to comparatively small sizes and the finer material not marketable, so that the remainder shall consist only of large pieces. To accomplish this thoroughly, it is evident that the coal should run freely over the bars, and that the bars should never become clogged up.

Bars made flat on top do not accomplish this object as thoroughly as pointed or rounded bars, because on a flat bar the fine fragments have no special tendency to run into the apertures between the bars.

Bars made with a diamond shaped (flat pointed) head, probably screen the coal better than any other form, and do not readily become clogged up; but when the coal is dumped from the car directly upon the bars, this form is not admissible, as it increases the amount of fine coal.

Bars with a rounded head (top) are probably better than any other form when the coal is dumped from the car directly upon the bars, but they are more likely to become clogged than pointed bars.

When the coal is run out upon the bars under a gate from a shute or pocket, the latter form is probably the best that can be adopted.

They are usually arranged in steps, as shown by Atlas Sheet No. XVI.,* so that the coal in passing from one set to the set below, falls one, two, or three inches. This fall jars the coal, the lumps roll over, and the dust and fine coal are shaken off.

Each set is made from three to five feet long, and the bars are seated on fluted plates, the fluting, or grooves, being about half an inch apart, so that the space between the bars may be changed at will, by seating them in grooves a certain distance apart.

They have generally been made of cast-iron, but wrought-iron and steel bars have lately been used. Ordinary rails without the bottom flange, make a very good form of bar for separating out the lump, but they do not answer as well for smaller sizes.

A thorough separation of the dirt and smaller sizes of coal from steamboat and lump, cannot be effected unless the coal is fed slowly and regularly. When a car load of coal is dumped directly upon the bars, a considerable quantity of small coal and dirt is sure to be forced over the bars upon the platform, unless the bars are very long; and even in this latter case, when several car loads are dumped in quick succession, the bars almost surely become more or less clogged up, and the separation is very imperfect.

These defects are almost entirely overcome by dumping the coal into a shute sufficiently large to hold two or three car loads of coal, and feeding the coal out slowly under a gate.

An improved form of screen bar, or rather a mechanical substitute for the screen bars, has lately been adopted by Mr. Eckley B. Coxe.

It is shown by Fig. 1 on Atlas Sheet No. XX, and consists of two frames, each carrying a set of narrow bars.

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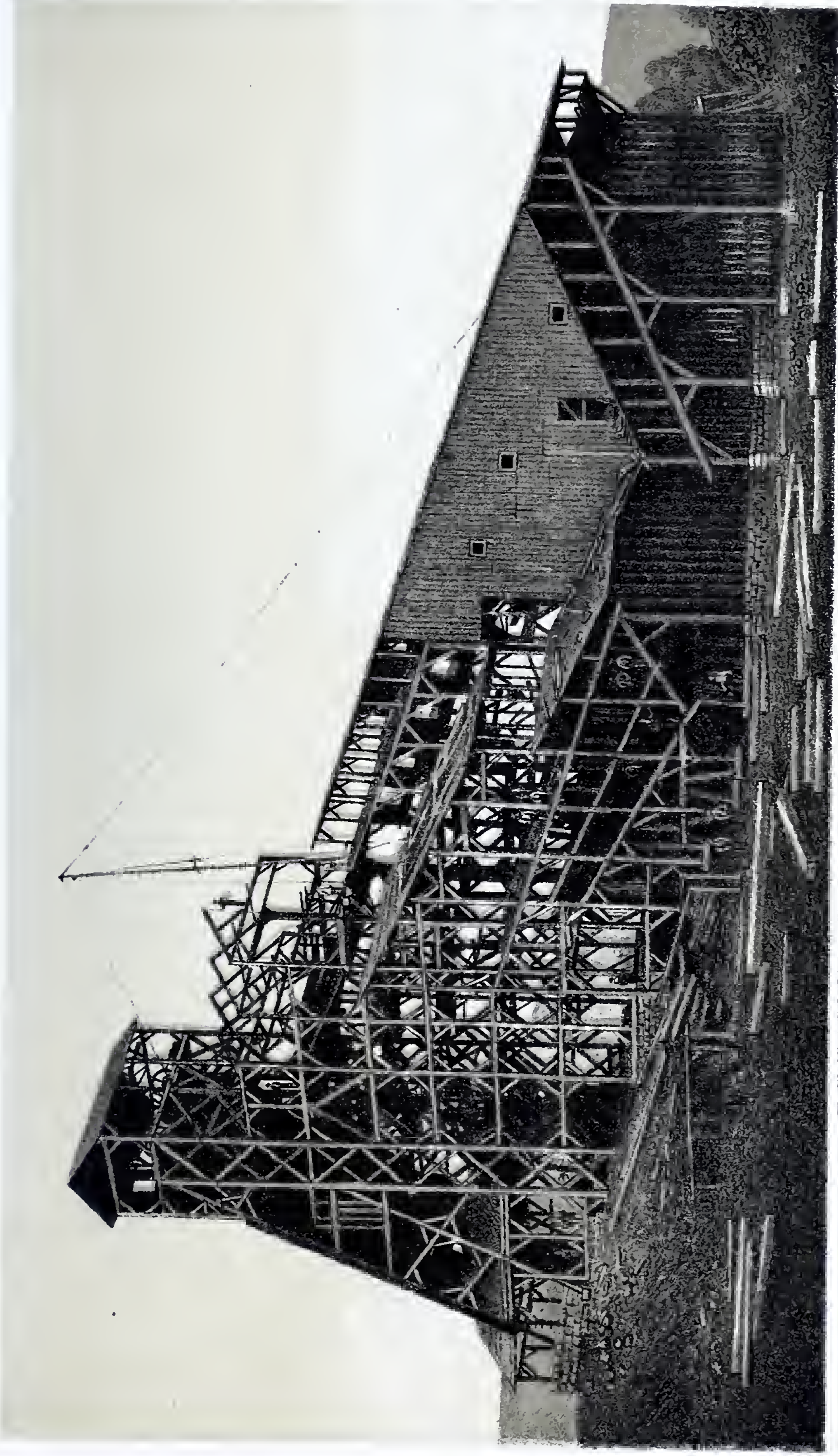
* See also Atlas Plate No. XVIII, C, C′, C″.



BREAKER IN PROCESS OF CONSTRUCTION

Each frame receives motion from the two shafts (which are connected by a rod not shown on the drawing) by eccentrics.

It will at once be seen that coal fed upon this apparatus at one end will slowly be transported to the other end when the frame is set horizontally, or even if inclined at a slight angle the coal will be transported forwards and delivered over the upper ends of the bars. At the same time the coal is slightly jarred, the dust and dirt shaken off, and all the small pieces are sure to fall through into the shute below.

The coal is dumped into a shute from which the bars feed it forward to the platform with perfect regularity. Whether the shute contains a small amount of coal or several car loads, the operation of these bars is the same.

Reciprocating screen bars of this description undoubtedly reduce the amount of labor to be performed on the platform, and better still, they distribute this labor evenly and regularly so that the men are not overcrowded with work when several cars are dumped in quick succession ; and, as the coal comes forward steadily,—not with a rush,—they can see at a glance the pieces of slate, of rock, of rough coal, and of bone, and handle them to better advantage.

The Platform.

This is a flat or slightly inclined floor upon which the large coal that has passed over the screen bars is cleaned and assorted by a number of men known as "platform men."

At large breakers the platform is often in reality a double platform, that is, it consists of two symmetrical halves. Thus there may be two holes to take coal to the crusher rolls, two divisions of the lump coal shute, two rock shutes, etc., but as the two usually form a continuous floor not separated by any partition or otherwise, it would not be well to make any such distinction.

The lump coal shute is generally immediately opposite the bars, there is a shute for rock slate and other refuse on each side, and one or two shutes or holes in the platform into which rough coal, fine coal made on the platform, and other coal that is to be broken into smaller sizes is thrown.

The platform men are expected to examine each large piece of coal and see that it is free from slate; lumps containing slate are broken with a pick or sledge, and the slate removed; and they are especially careful to allow no coal containing slate to go to the rolls.

When the rough coal, etc., is thrown into the crushers through holes in the platform, these holes are located within three or four feet of the bars, and are provided with a hood or guard to prevent the lump coal from falling into the rolls.

Lump and Steamboat Shutes.—These answer in place of pockets and generally (especially the former) hold a very large quantity of coal. They extend from the level of the platform,—the steamboat shute from a lower level,—on a slope of say eighteen degrees, to within eight or ten feet of the level of the railway track.

Rolls.

Page plate No. 47 shows a pair (or set) of rolls used for breaking coal. Rolls of various forms have been introduced at different times but the plain cylindrical crusher with cast-iron or movable steel teeth is about the only kind of crusher now used.

It has been clearly proven in practice that the rolls now made with movable steel teeth curved forward (known as the "hawk-bill tooth) make from two to six *per cent.* less fine coal than the old style rolls with cast-iron pyramidal teeth.

The teeth now used are four-sided and have the front edge almost straight, but the back is curved so that the *points* of the teeth strike the lumps and draw them into the rolls, splitting them up at the same time.

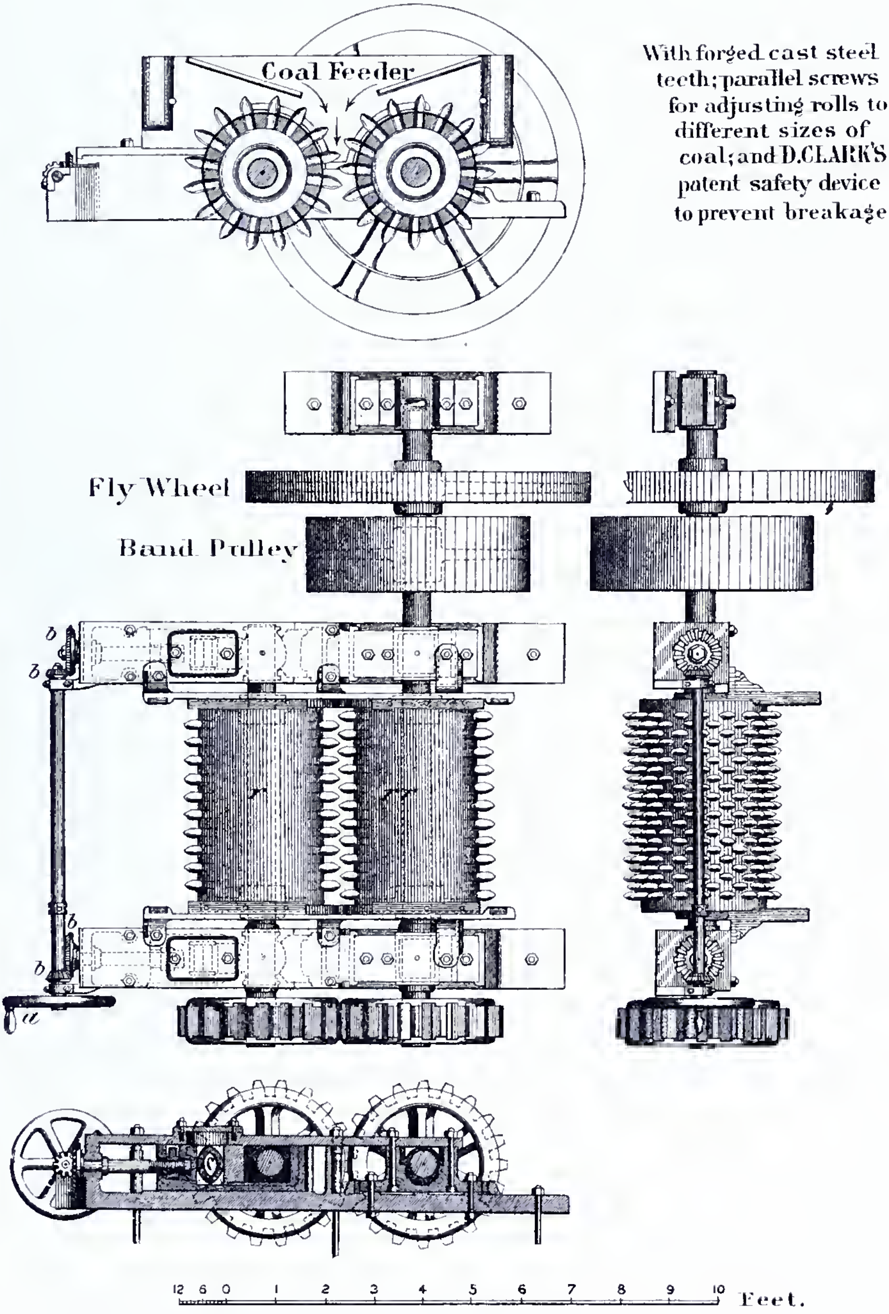
It is also known that these rolls make much less fine waste when they are new, and the teeth are sharp, than after they become dull.

Experiments made by some of the operating companies have also shown that the waste made by the rolls may be

STEEL TOOTH BREAKER ROLL,

Built by D. CLARK & CO.

HAZLETON, PA.



With forged cast steel teeth; parallel screws for adjusting rolls to different sizes of coal; and D. CLARK'S patent safety device to prevent breakage

increased as much as *five per cent.* by overcrowding them with coal, or feeding coal faster than the rolls can take it.

It has also been proven that the amount of waste is less when the coal is not broken at once into small sizes, but is passed through two sets of rolls,—the second set being closer together than the first set,—but the saving effected in this way is small.

The size of the rolls has some influence on the amount of waste made, and large rolls are now preferred by many colliery managers.

The illustration (Page plate No. 47) shows a safety device by which breakage is prevented when the rolls draw in material too hard to crush without breaking either the teeth, rolls, or gearing.

The frame carrying the bearings rests against a shell of cast-iron inserted in a box. This shell is made thick enough to stand a thrust of not more than nine* tons (?); if subjected to a greater pressure it breaks and allows the rolls to slide further apart, relieving them of all strain..

The cast-iron shell is quickly replaced by removing the cap covering the box in which it is placed.

The illustration also shows a convenient form of gearing by which the distance between the teeth (the two rolls) can be varied at any time, even when the rolls are running full speed.

"Crusher" or "Steamboat rolls" are now usually made from 2′ 9″ to 3′ 6″ in diameter and about three feet long. These are also sometimes called "Prepared coal rolls," and the smaller rolls "Pony" or "Monkey" rolls,—but the largest rolls are generally known by the former names; the next size smaller are called "Prepared coal rolls," and rolls used for breaking down egg or large stove to smaller sizes are known as "Monkey rolls" or "Small monkey rolls."

Screens.

These are always cylindrical, and are inclined at a slight angle so that the coal will slowly travel from through the screen from one end to the other. The inclination is usually about one inch in twelve. Increasing the pitch has the same effect on the screening as shortening the screen, but increases instead of diminishes its capacity. Hence, the screening capacity in large breakers is increased by increasing the pitch of the screens, but to secure thorough screening very long screens are necessary, while at breakers handling a small output the screening may be equally well done by short screens with a very small inclination.

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*This figure may not be correct; I am not certain that I remember it correctly.—H. M. C.

The *main screens* are usually from twenty to thirty feet in length and are made to separate three or four sizes; they are also called *stove screens.*

The broken screens, the mud or dirt screens, and the chestnut or pea screens are commonly from eight to sixteen feet long,—the broken screens are sometimes made twenty to twenty-four feet long.

In diameter they range from four to eight feet.

They are constructed of a number of cast or wrought-iron spiders set at intervals of from three to five feet on a wooden, a cast or wrought-iron shaft, and covered in between with wire or cast-iron gratings of the proper mesh.

The spiders naturally divide the screen into "segments," and each segment may have a mesh of different size, but it is generally found necessary to make at least two adjoining segments of the same size.

The segments at the upper end of the screen, or the end into which the coal is fed, carry the smallest mesh and those at the lower end are made of the largest mesh.

Cast-iron meshes are now greatly preferred to those woven of wire or small iron rods, except for the very smallest meshes. Small meshes made of cast-iron are very weak, and cannot stand the blows given with a hammer from time to time, to shake out the fine dirt with which they become clogged.

Phœnix columns are now used by some companies for screen shafts. Wooden shafts are too elastic, and when long they spring in the center and break the segments, (when made of cast-iron); cast-iron shafts are too heavy, and wrought-iron pipe of large size is more expensive, and at the same time is not as stiff as a Phœnix column.

I do not know that any attempt has yet been made to run the screens on outside friction pulleys, but it seems probable that such a plan might give satisfactory results, and do away with these heavy shafts.

The peripheral speed at which screens are run varies from about one hundred and twenty to two hundred and twenty feet per minute, but we generally find them running at a speed of from one hundred and sixty to two hundred feet per minute.

Short screens are frequently made with an outside jacket carrying a finer mesh than the main body of the screen, and long screens are commonly jacketed at the upper end.

The outside jacket is commonly made of a very small mesh to separate the fine dirt from the chestnut, pea, and buckwheat sizes, or from the two latter sizes, so that when this coal goes to the chestnut or pea screen, a considerable quantity of dirt has already been separated from it. In other cases the jacket is made with a large mesh ($2\frac{1}{2}''$ or $1\frac{7}{8}''$) to take out from the end of the jacket the egg or large stove coal, the main body of the screen separating out the broken. In this case all the smaller sizes pass through the jacket (mesh) to go to the stove or main screen.

Chestnut and pea screens are nearly always jacketed; the dirt passes through the outer mesh, the buckwheat passes through the lower segment of the jacket, the pea out of the end of the jacket, and the chestnut out of the end of the inside screen; or the buckwheat comes out at the end of the jacket, and the pea coal comes out at the end of the inside mesh.

Two methods of driving screens are in common use:

1. By bevel gear on the screen shaft, usually at the lower end of the screen, as shown by Atlas Sheet No. XVI; or,

2. By cog-gear on the periphery of the screen, at its upper end, as shown by Atlas Sheet No. XVII, and by the main screens at the Hollenback Breaker, Atlas Sheet No. XVI.

The slate picking segments have already been described.

They are usually narrower than the regular screen segments, and are useful only when a considerable portion of the refuse occurs in flat pieces, and the coal does not break in this way.

To insure proper screening, *i. e.*, the removal of all the fine dirt, and satisfactory sizing of the coal, with a minimum amount of loss, it is absolutely necessary to feed the screens regularly.

When the screens are crowded with coal, we always find a considerable quantity of dirt remaining in the coal, and we also find chestnut and pea coal mixed in with the stove, stove mixed with the egg, etc.

The screen feeder shown by Atlas Sheet No. XVII, as used by the Lehigh Coal and Navigation company, removes this difficulty; but appliances of this kind have not been generally adopted. The illustration plainly shows its action, so that description of it is not necessary.

When the coal is wet and dirty, washing is necessary to remove the dirt that adheres to each fragment. This is accomplished by a series of small streams or jets of water falling upon the coal in the screens from a perforated pipe or trough above the screen. The coal coming down over the main screen bars is sometimes washed in the same way.

This washing is not only necessary because without it the coal would look very dirty and not find a ready sale, but also because the coal must be bright and clean to enable the slate-pickers to readily distinguish the slate, bone, etc., from good coal.

It is not possible by screening to separate the coal into grades of even approximately uniform size, hence we find that in any one size of coal the largest pieces will weigh two, three, or four, (or more) times as much as the smallest fragments. This is because the coal breaks into pieces of almost every conceivable shape.

Thus in "large stove" coal screened over a $1\frac{7}{8}$ mesh and through a two-inch mesh,—which is very close screening,—a flat triangular piece of coal about $2\frac{1}{3}''$ in its least width and say one half inch thick, is the smallest piece that will not pass through the $1\frac{7}{8}''$ mesh; it will contain about $1\frac{6}{10}$ cubic inches. The largest piece that will pass through the 2″ mesh will be "say" 1.9 inches square, but may be three, four, or more inches long; if it is three inches long it will contain about $10\frac{8}{10}$ cubic inches. In practice we usually find that the largest pieces in any one particular size will weigh from two to five times as much as the smaller pieces. This great variation in size, shape, and weight is one of the greatest difficulties in separating the slate and other refuse by jigging.

Jigs.

Many different styles of jigging machines have been used, but only a few have given satisfactory results. The Bradford and the Clark jigs (see page plate No. 48,) are probably more favorably known than any other forms.

Jigging should remove all of the slate, rock, and sulphur balls, but no machine has yet been invented that thoroughly accomplishes this work. The reason why jigging as commonly practiced in the anthracite coal-fields does not thoroughly clean the coal, is probably not because the machines are unable to do the work, but because the jigs are often overcrowded (fed too fast) and the coal to be washed consists of fragments varying largely in size.

To secure a complete separation of the refuse from the coal, it is absolutely necessary: 1. To have the material to be jigged of nearly uniform size and shape, and 2. To feed the jigs slowly and regularly.

Before being jigged the coal is separated into the various commercial sizes (egg, large stove, small stove, chestnut and pea) by passing over a certain mesh and through a larger mesh, but screening in this way does not produce coal of uniform size. If the pieces of coal were all spherical the pieces of coal in each size would be of approximately uniform weight, but as anthracite breaks into fragments of almost every conceivable shape, we can find in any size of coal pieces that weigh three or four times as much as the smallest pieces. It therefore follows that sizing by screening does not satisfy the first requirement. Again, much of the slate occurs in flat pieces that are easily buoyed up by the water and pass over with the coal.

Jigging coal in the anthracite regions is rarely a cheap process. At a large number of mines the coal must be washed and jigged with mine water strongly acid, and the parts exposed to the action of this water are rapidly corroded, so that repairs are constantly needed.

In the diagram showing the method of preparation at the Lehigh Coal and Navigation company's breaker No. 10 it will be observed that the coal sized by the mud screen is jigged before being broken by monkey rolls. This method of preparing the coal (a double process for this portion of the coal) is necessitated by the large quantity of small slate fragments and other refuse in the coal as it comes from the mine.

When the coal comes from the mine in a wet and dirty condition, the dirt adhering to the coal is commonly washed off by jets of water. The water is thrown upon the coal in the screens, and upon the screen bars, from wooden troughs or perforated iron pipes which are located a few feet above the screens.

The cost of washing and jigging coal is greatly increased by the necessity of using mine water for these purposes. The screens, jigs, shute and pocket linings, and any iron with which the water comes in contact, are quickly corroded and need constant repairs.

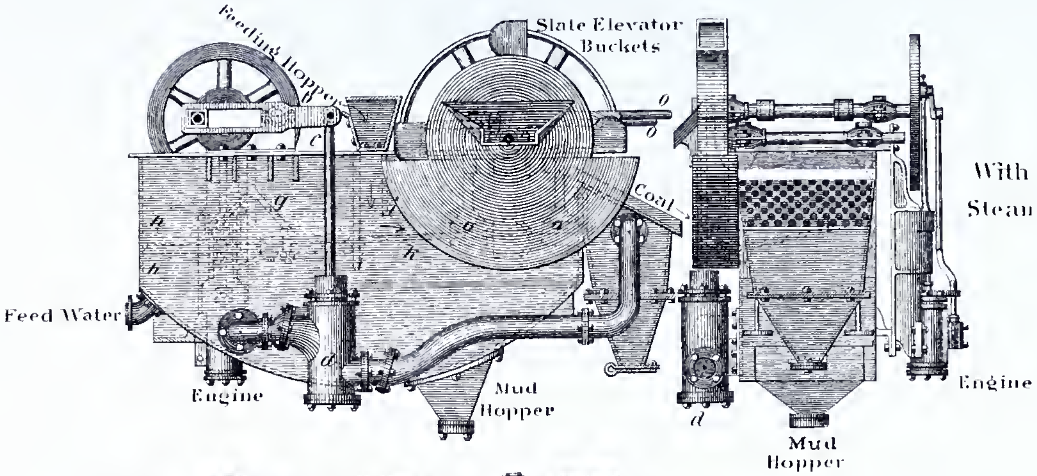
Picking Tables and Shutes.

The picking shute, or "telegraph," in common use, is an ordinary trough inclined at a sufficient angle to allow the coal to slide down towards the pockets. Boys, and old men too feeble to perform hard labor, are stationed along one or both sides of this trough to pick out the fragments of slate.

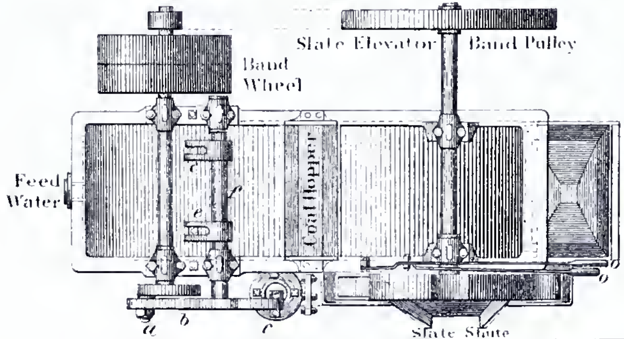
The slate picker frequently sits astride the shute on a board seat, and controls the flow of coal with the heel of his boot, or with a short stick placed diagonally across the bottom of the trough.

It is evident that more labor is required to separate the slate by this method than would be required if the coal

THE CLARK JIG FOR SEPARATING SLATE AND ROCK FROM COAL.



With small Vertical Steam Engine attached



Without Engine. Power taken from breaker shafting.

NOTE: This Jig was patented July 1st 1879.

Built by D. CLARK & CO. Hazleton Pa

was not in motion, as many pieces of slate and bony coal must inevitably slide past unobserved by the slate picker, or may be beyond reach when seen.

The picking table is not open to this objection, as the coal runs out on a nearly level platform, (long and narrow,) and must be pulled over the edge of this platform by hand ; it then drops into a shute leading to the pocket. But the labor of drawing the coal forward to the edge of the table by hand, although the distance is very small—from six to twelve inches, often less—is an objection always urged against this plan.

The difference between these two methods may be briefly stated: When the coal is cleaned in picking shutes it is imperfectly examined for refuse by several slate pickers in succession ; when cleaned on a picking table it is examined thoroughly once, by only one slate picker.

A combination of the picking shute and picking table has lately been invented by Mr. F. B. Parish. It is shown by Fig. 4 on Atlas Plate No. XX. The coal runs down the shute, which is slightly inclined towards one side, (or is elevated along the center line,) so that a portion of the coal runs in upon each one of the diminutive picking tables placed on one or both sides of the shute. A hole in the shute allows the coal to pass through into a shute leading to the pocket, as shown by the cross-section. Any further description is unnecessary, as the drawing (although faulty in perspective) shows the arrangement clearly enough.

Elevators.

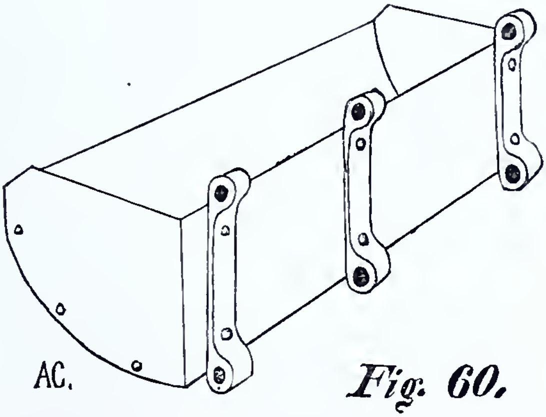


Fig. 60.

Mention has frequently been made of the elevators sometimes used for raising coal in the breaker. They are similar to elevators used for raising ore and heavy materials in other mining districts. One form of bucket is shown by Fig. 60. These buckets, when connected together by iron bolts or rods, form an endless chain.

Breaker Engine.

Any good stationary engine is suitable for driving a breaker. The power is commonly transmitted through belt gearing, and the engine is subjected to no greater strains than engines used for driving other kinds of machinery. The location of some breaker engines at some distance from the breaker, the power being transmitted by a wire rope, has already been mentioned.

Breaker engines are often old second-hand engines already owned by the operator, or bought for that purpose.

Breaker Ventilation.

When the coal is prepared dry, the breaker is generally so clouded with dust that, notwithstanding the fact that one or two sides of the breaker may be riddled with windows, or almost entirely composed of glass, it is impossible for the slate pickers to readily distinguish coal from slate or bony coal.

The page plates illustrating this report unfortunately do not show any well lighted breakers, but some of these structures contain an enormous amount of glass. To thoroughly prepare the coal at minimum cost, it is absolutely necessary to have good light.

It is evident that we cannot expect to obtain the requisite amount of light while the breaker is clouded with dust, even if we build the whole structure of glass. We must, therefore, get rid of this dust.

Mention has already been made of the ventilation of some breakers by small exhaust fans. I think I can safely assert that this plan could be adopted with good results at all breakers preparing dry coal, and that the day is not far distant when we shall see a fan at work in all such breakers.

CHAPTER XXVIII.

Waste in Mining and Preparing Anthracite.

The waste in mining and preparing anthracite coal for market is naturally divisible under two heads:

1. Waste in mining.
2. Waste in preparing and handling.

The records kept at the breaker generally include in the "waste," the slate, rock, and bony coal which is separated from the merchantable coal in process of preparation, thus largely increasing the apparent percentage of "waste;" but it is evidently unfair to include as waste any material except *fuel* thrown aside as refuse.

The waste in mining includes not only the coal left in pillars to support the overlying strata, coal lost by a "crush" or "squeeze," and coal left in chain pillars, but should also be considered to include all the fine unmerchantable coal (culm or coal dirt) made in mining and in transporting the coal to the breaker.

The waste in preparing should be considered to include only the fine coal (culm) made in breaking, screening, handling, and loading, and the coal adhering to lumps of rock or slate sent to the rock dump.

It is difficult to determine exactly the amount of waste made by the rolls in crushing the coal. Every set of bars, every screen, shute, and jig through which the coal is passed increases the waste, making it extremely difficult to discover what percentage is actually caused by breaking the coal.

It is equally difficult to determine the exact percentage of dirt (fine coal) made in mining and in transporting coal to the breaker.

This subject has already received attention by the Survey, and I shall not attempt to add to the large mass of material published in Report A^2, by Mr. Franklin Platt.

Without quoting in detail from that report, I shall endeavor to summarize some of the conclusions that may be drawn from its facts and figures, and to give some averages deduced from the tables of "breaker waste."

Many of these tables (in Report A^2) show an enormous percentage of "dust" or "dirt" going to the culm banks, but it should be clearly understood that under this title are often included slate, rock, sulphur, and other material rejected in preparing the coal—the term "*refuse*" would have been more applicable,—so that while some of the tables show from *thirty to fifty per cent.* of "dust" going to the refuse heap, it may at the same time be true that only a much smaller percentage is actually *waste*,—*i. e.*, fine coal,—and the remainder is *refuse*, and not fuel.

The amount of coal wasted by being left in the ground is influenced by:

1. The thickness of the bed;
2. Its dip, whether steep or flat;
3. The hardness of the coal;
4. The depth at which it is mined;
5. The character of the roof, and (on steep dips) of the floor.

Thus this waste is greatest in a thick bed of soft coal mined at considerable depth; and least in a thin bed of harder coal mined at a small depth with good roof, or at greater depth with a weak roof.

In the anthracite regions it is variously estimated in different localities and under varying mining conditions, as shown below.

In the Lehigh region Mr. T. D. Jones estimates the amount *left in the ground* as follows:

Mammoth,	28′ thick,	30 to 40 per cent.
Wharton,	8′ "	10 to 20 " "
Buck Mountain,	12′ "	20 to 30 " "
Summit Hill Mammoth,	60′ "	45 " "

Greatest waste on dip of 45°, least waste on flat or vertical dip. His estimates of waste appear to be very low.

Mr. Eckley B. Coxe says "The average yield is at least 10,000 tons per acre,—or *1,000 tons per foot per acre*,"—which corresponds to a loss of about (47?) per cent.

Col. D. P. Brown estimates the amount lost in mining the Mammoth bed in the Lost Creek (near Shenandoah) district as follows:

Dip 60° to 80°, good roof,	30 *per cent.*
" 20° to 60°, " "	40 " "
" 60° to 80°, poor top, coal shelly,	50 to 55 " "
" 20° to 40°, " "	50 to 70 " "
" 15° to 35°, panel system,	20 " "

In the Wyoming region the amount lost by being left in the mine is less than in the Western Middle coal field.

Until we have more careful measurements than have yet been made, it will be impossible to estimate the percentage of coal lost in pillars, etc., in the mine.

From my own knowledge of the methods of mining throughout the region, I believe that this loss is at least
35 (to 50) per cent. for thick seams with good roof;
25 per cent. for seams twelve feet and less with good roof;
15 " " for thin seams of clean coal with good roof;
40 " " for seams with poor roof;
55 " " for thick seams with poor roof;
and that the amount lost in pillars and by crushes will average at least 45 per cent.

These figures embrace only the coal left standing in the mine, or ruined by a squeeze, and to them we must add a certain percentage that is lost by being left in the mine as dirt, (culm,) or adhering to lumps of refuse, or so intimately mixed with slate, rock, and other refuse, that it does not pay the miner to separate it. This will increase the *average* percentage of coal left in the mine to *at least 50 per cent.*

In some districts where seams of moderate thickness, containing little refuse, and with a good roof, are worked on comparatively flat dips, the amount left in the mine is much less than 50 *per cent.*, but with thick, impure seams, with a poor roof, the loss is much greater.

The second division of mine waste,—the fine coal made in mining and transporting the car to the breaker,—varies with, 1. The character of the coal; and, 2. The dip of the bed.

The amount of fine coal made in mining depends principally upon the softness or brittleness of the coal. It is increased by the use of unnecessarily large charges of powder, but as the miner pays for his own powder, he is not likely to use more than he thinks necessary to dislodge the coal.

The waste from this source varies from four or five to fifteen per cent. or more of the quantity mined. At some collieries the amount of fine coal is enormous, but this is owing to the fact that the coal exists in the bed in a squeezed or "shelly" condition, (or even as dirt,) and cannot properly be charged to "waste;" it is actually waste or refuse already present as such. The above figures are intended to apply to the amount of fine coal made in mining from a compact, unbroken bed.

It will be safe to assume an average of at least from five to ten *per cent.* as the waste from fine coal made in mining and transporting the coal to the breaker.

When the coal is flat enough to take the wagon in to the face, the waste is much less than in highly inclined workings, in which the coal slides or falls down a shute, from the face to the gangway.

When the coal is steeply inclined and being worked "by the run," all the material (coal and refuse) is drawn from the breasts and sent to the breaker, the waste from coal becoming mixed with refuse* is avoided, but a considerable quantity is still lost by adhering to lumps of slate that are sent to the rock dump. An inspection of any rock, slate, or culm banks will at once furnish conclusive evidence that the quantity wasted in this way is not small.

Again, recent analyses have shown that some of the material rejected as refuse—that known as bone—frequently contains a large percentage of carbon and yields only a moderate amount of ash, being in fact a fuel of no mean value, but as its dull slaty appearance decreases the market value of the coal it is rejected and sent to the waste banks.

I estimate the waste from the above-described causes, thus:

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*See a preceding page.

1. Coal left in pillars and lost by being crushed etc.,	45	per cent.
2. Coal lost (in mine and at breaker) by adhering to or becoming mixed with refuse,	7	"
3. Fine coal, made by blasting, running through chutes in mine, loading, etc.,	8	"
Coal wasted before reaching the breaker,*	60	"
Coal reaching the breaker, (excluding fine coal,)	40	"

Assuming the waste at breaker from breaking, at 10 *per cent.*, from screening, at 4 *per cent.*, and waste at lip screens, at 2 *per cent.*, making a total of 16 *per cent.*, we have:

Coal reaching breaker,	40	*per cent.*
4. Breaker waste, (16 per cent. of 40,)	6.4	"
Coal loaded into cars and sent to market,	33.6	"

This percentage (33.6) has not been manufactured to sustain the common statement, that but one-third of the coal in the ground is sent to market, but is the result unexpectedly obtained after much careful study of the data contained in report A², supplemented by my own knowledge of the mining and preparation of anthracite. Any change that I could make in these estimates would reduce, rather than increase this percentage.

Few accurate measurements of worked out areas have yet been made, and until this is done it will not be possible to arrive at the exact percentage.

Estimates at the Locust Run, Staunton, and Gilberton collieries show that 33.5, 17.4, and 24.6 *per cent.* of the coal in the ground has gone to market, and the average of six measurements, by Mr. I. A. Stearns, of beds from 5 to 11 feet thick in the Wyoming region, show that 44.8 *per cent.* has been shipped.

These figures give an average of 30.1 *per cent.* shipped to market and 69.9 *per cent.* of waste.

I think that we may safely conclude that our present methods of mining and preparing coal involve the loss of from 66 to 72 *per cent.* of the total amount of coal in the ground, and that from 28 to 34 *per cent.* is shipped to market.

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*This includes coal lost by adhering to lumps of refuse rejected after the coal has been dumped and in process of preparation in the breaker.

Breaker Waste.

Only the fine coal made by breaking, screening and handling the coal should be included under this head. It is divisible into

1. Waste made by breaking,
2. Waste made by screening,
3. Waste made by passing through shutes to different parts of the breaker, in falling into the pockets, and in loading.

As the waste made in loading is necessarily shipped to market we will ignore it; the waste made by shutes, etc., is separated by the lip screens and is usually spoken of as "lip screen waste." The waste made in screening is partly removed by the dirt meshes, and partly by the dusting bars placed beneath the screens, but a portion of this waste always finds its way to the lip screens.

The percentage of waste made by the rolls depends upon

1. The character of the coal,
2. The percentage to be broken into sizes for domestic use,
3. The style of rolls used, their speed, etc.

At the Ashley No. 6 breaker of the Lehigh and Wilkesbarre coal company the waste made in breaking coal from the Baltimore bed was 11.56 *per cent.*, from the Ross bed was 10.99 *per cent.*, but coal from the Red Ash bed made only 5.88 *per cent.* of waste.

Mr. Howard Chester estimates the breaker waste in preparing coal from the Lyken's Valley bed at 19 *per cent.*, while Col. Brown has found the waste to be 15 *per cent.* in breaking and preparing coal from the Mammoth bed in the Shenandoah district. The above figures clearly show how variable is the percentage of waste due to differences in the character of the coal. The percentages apply to the amount *broken*, and not to the quantity shipped.

At collieries mining coal marketed in large sizes (lump, steamboat and broken) for blast furnace use, etc., the percentage of breaker waste is of course very much less than at mines working a coal marketed almost exclusively for domestic use.

The amount of waste going to the culm bank also varies with the quantity of buckwheat coal (if any) used at the colliery, or marketed, and with the size of mesh used for separating the culm and buckwheat.

Buckwheat coal is still commonly treated as waste, but only a few years since, coal of the size now marketed as pea and even chestnut were discarded as refuse, and sent to the culm banks with the dirt and slate.

The data given in Chapter VII of Report A, demonstrate the large reduction in breaker waste resulting from the improved steel-tooth rolls. They show a reduction in different experiments as follows:

	Old Style Rolls. *Per cent. Waste.*	*New Style.* *Per cent. Waste.*
Empire breaker, Baltimore bed, . .	11.96	8.03
" " Hillman bed, . . .	17.68	11.96
Sugar Notch No. 10, ...	18.75	13.15
Diamond Breaker, ...	. . .	11.88
Lance No. 11, ...	15.90	
Nottingham No. 15, ...	12.36	
Reynolds No. 16, ...	17.06	
Ashley No. 6, Baltimore bed, ...	. . .	11.56
" " Ross bed, ...	. . .	10.99
" " Red Ash bed, ...	. . .	5.88

Col. Brown estimates the waste made in breaking the Mammoth bed in the Shenandoah district at 11.27 *per cent.* of the coal broken, and the loss from screening, and by shutes, loading, etc., at 6 *per cent.*

Two experiments made by him show the breaking and screening waste to be 15.84 *per cent.*, and 14.7 *per cent.* or an average of 15.27 *per cent.*; of which he considers 11.27 *per cent.* caused by breaking.

Other experiments made in the same district gave:

Breaking Steamboat to Broken Egg, etc., ...	20%	waste.
Breaking Broken to Stove size, etc., ...	25½%	"
Breaking Egg to Stove size, etc., ...	14.4%	"

The waste includes all that passed through a ⅜ inch mesh.

In the Shamokin district the waste from breaking and screening is estimated at some collieries to be as high as *25 or 30 per cent.*, and at a large number of collieries the waste is certainly twenty per cent. or more. We may divide this approximately into

Waste from breaking,	14 *per cent.*
Waste from screening, etc.,	6 *per cent.*
	20 *per cent.*

About the same percentage of waste from breaking and screening is made in the western half of the Southern coal field.

The hard coals of the Eastern Middle, (Hazleton, etc.,) coal field produce much less waste. Mr. Thos. McNair gives:

Mt. Pleasant colliery,	13.50	per cent. waste.
Hazleton No. 6, "	12.80	" "
Hollywood colliery,	14.36	" "
Harleigh "	8.35	" "

As a result of careful experiments made to determine the waste made by breaking by hand, screening by hand and by circular screens, Col. Brown gives the following figures:

Waste made by breaking down to egg and stove size by hand with small hammers, the coal then being screened by hand,	6.28 *per cent.*
Waste made by running this coal through circular screens,	4 *per cent.*
Waste made by breaking by hand and screening in circular screens,	10.28 *per cent.*
Waste made by breaking with rolls and screening in circular screens,	15.27 *per cent.*

From the above we learn that the waste made by breaking by rolls was 11.27 *per cent.*, (15.27 — 4.00,) and by hand 6.28 *per cent.*, so that the method of breaking with rolls produced only 5 *per cent.* more waste than by breaking by hand. When the coal is very soft and easily shattered into dust this increase in waste by crushing over hand breaking may be much greater,—say from seven to nine *per cent.* of the amount broken.

Reduction of Waste.

As the waste has been shown to be caused principally by the large quantity of coal left in the mine, our attention is naturally diverted from the breaker, which really produces a small percentage of waste, although it is frequently charged with much more waste than can fairly be attributed to it.

Next in importance to the coal lost by being left to support the roof, is the coal lost by becoming mixed with slate and other refuse in the mine (in flat workings); that lost either in the mine, or at the breaker, by adhering to lumps of refuse material, and the fine coal made in blasting and transporting the coal to the breaker.

It is evident that we cannot expect to effect any great reduction in the waste made by the breaker. Screening and handling is unavoidable, and the improved rolls now used do not make an unreasonable percentage of waste. Nor can we expect to greatly reduce the amount of fine coal made by blasting and by transporting the coal to the breaker.

The coal wasted by being rejected with lumps of refuse to which it adheres, or by becoming mixed with refuse (gob) left in the mine can undoubtedly be diminished. The amount so wasted will always be governed by the ruling market price of coal. In other words, when the price realized for the coal is high enough to allow the operator to have all the coal carefully separated from the refuse, at a profit, the waste from this cause will be reduced; but no operators can be expected to recover this coal when the cost of separation is greater than its money value.

We are therefore forced to look forward to the adoption of improved methods of mining as the only means by which the percentage of waste can be *largely* decreased, and also to a more careful separation of the coal and refuse.

Assuming the adoption of some method by which all the coal might be recovered, it may be of interest to inquire how much could be shipped to market as merchantable coal.

With our present system of blasting and preparing the coal, we would waste (on a basis of 15 *per cent.* of lump and steamboat) in fine coal:

By blasting and handling, say,	10	*per cent.*
By breaking and screening, (16% of amount broken,)	12	"
Waste,	22	"
Percentage merchantable coal,	78	"

If all the coal is broken for domestic use, we have:

By blasting, etc.,	10	*per cent.*
By breaking, etc., 16%×90,	14.4	"
Waste,	24.4	"
Percentage merchantable coal,	75.6	"

If we could mine 90% of the coal in the ground, we could ship 67½%; mining 80%, we could ship 60%; mining 70%, we could ship 52½%; mining only 60%, we could ship 45% of merchantable coal,—assuming always a thorough separation of the coal from the refuse.

The quantity of material lost or considered as wasted, depends, of course, upon the size of the smallest or finest coal that can be burnt as fuel. This will be considered further on. It is, therefore, evident that improved mining methods are absolutely necessary if we are to reduce the enormous wastage that is every year sacrificing two tons of fuel for every ton sent to market.

It must not be inferred that no progress has been made in the past, for the amount of coal now recovered is often nearly twice as much as would have been mined from the same bed thirty years ago. And progress is still being made, and although no radical changes have or seem likely to be effected, the adoption of improved methods is steadily going on.

It will be in place here to consider some of the proposed improvements in our mining methods, as well as those that have been tried and more or less generally adopted.

1. The boundary plan of leaving thick pillars to divide the workings into panels, so that a squeeze induced by robbing the pillars out in any part of the mine is localized to that panel. Thick pillars should be, and now generally are, left along property lines, to isolate and protect the workings on one side from fire, flooding, or a squeeze occurring in the adjoining mine.

2. *The plan by which narrow workings* are first driven, leaving the pillars of extra thickness until the mine is exhausted and robbing commences, when the pillars are either robbed by skips taken off the rib, or by opening through the center of each pillar and robbing back. This plan decreases the profits as long as the workings are advancing, but during the time of robbing back the returns will be larger. In many localities the percentage of coal recovered could be increased at least ten, and perhaps twenty *per cent.* by the adoption of this plan. It has been adopted at a few collieries, but as yet has not had a fair trial.

This plan is now in use at the Lehigh colliery (L.V.C.Co.) on the Girard lands, where a breast is opened at every sixty yards on the gangway, so that a solid pillar of coal fifty yards thick is left between each two breasts, the breasts being ten yards wide. The pillar thus left is wide enough to open two breasts each ten yards wide with ten yard pillars, but these breasts are not to be opened until the gangway is driven to the limit. Work will then be commenced in the pillars farthest from the outlet (nearest the limit of working) by opening two breasts in each, and robbing out the plllars as soon as these breasts are worked up to the gangway above, following this up by robbing out the gangway.

It will be observed that in this plan of pillar and breast working, only every third breast is opened as the gangway is being driven; in other localities it is proposed to adopt the same general plan, but to open every alternate breast in driving, thus leaving a thirty yard (more or less) pillar between each two breasts.

I understand the same plan is also being tried by the Philadelphia and Reading Coal and Iron company.

3. *The plan of driving to the limit* before opening any workings, and then opening up the coal and working back, only a few breasts being opened at once, and these worked as fast as possible. Many engineers admit that by this plan from ten to twenty-five *per cent.* more coal could be

obtained, but as it necessitates a large outlay at the start, it has not received favorable attention. This plan is really identical in principal with the preceding method, but the charge it involves is more radical. I think we may confidently anticipate its more or less general adoption in the near future.

4. Veith's boundary plan, in which the pillar and breast stumps above the gangway are left double or more the ordinary thickness, so that the pillars may be robbed without danger of closing the gangway. It is now being tried by the Philadelphia and Reading Coal and Iron Company.

5. Brown's panel system, which has been in use for a number of years by the Lehigh Valley Coal Company at Lost Creek. Col. Brown finds it especially applicable to mining thick seams on dips of from 15 to 35 degrees. It is undoubtedly more costly than other methods, and this is probably the reason why it has not yet been adopted by others.

6. The Rock-shute plan of opening the gangways and airways in an underlying (thin) bed, and tapping the thick bed by rock shutes. When a bed occurs in the right place beneath the Mammoth or any other thick bed, I think the adoption of this plan will always give satisfactory results. In many cases from ten up to thirty or forty *per cent.* more coal could be recovered. When the lower bed is not more than thirty feet distant and the intervening rocks are not very hard, this method will probably be found to be as cheap as the plan now used. It seems probable that we shall hear more of this method in the future.

7. Long-wall and panel workings in comparatively flat workings on beds of moderate (ten or twelve feet or less) thickness. Whether the long-wall system can ever be successfully applied to anthracite mining is a problem to be decided in the future. By limiting the length of working face to one or two hundred feet, and working in panels, *by withdrawing*—never advancing—it is not improbable that from seventy-five to eighty *per cent.* could be mined under certain conditions. Conditions favorable to the success of this method are present in parts of the Wyoming coal field, but we cannot expect either its trial or adoption at present.

It involves a large amount of dead work before any coal can be won, and consequently a large outlay of capital at the start. Whether the cost per ton would eventually be any greater is questionable. In gaseous districts opening by this plan would be a very slow method of development, and much more dangerous, *at first*, than the plan commonly pursued.

8. "*Remblais*," or packing the breasts full of mine refuse and material taken from the surface, so that the pillars may be taken out, is a method that has been used in the coal fields in the south of France, and has, I believe, been tried at the Midlothian mines in Virginia. It does not seem probable that any such system will be successful in the anthracite districts, at any rate not for many years to come, and it does not offer much promise of success in dealing with thick seams.

9. What may be called a combination of long-wall and " overhand stoping" is in use abroad, in mining very thick beds on moderate dips, (15° to 30°.) Shutes are driven up from the gangway until they are nearly (or quite) holed through into the gangway above, and a long face opened out. The coal is then taken out in slices or steps, the top bench being worked back first and the roof allowed to come down, a second bench is then mined, and finally the bottom bench is taken out. When the benches are left, forming long steps, the roof is supported by a series of props on each bench, so that it gradually settles or arches down to the floor, and the whole bed is removed at once. It is evident that this system can only be successful when the roof is a compact, homogeneous, tough, and elastic rock. It is undoubtedly a very dangerous method, and the cost of the dead work and of the props is not small. I am very doubtful whether there are any beds in the anthracite region that could be successfully worked by this plan.

While it would certainly increase the output, (if it could be used at all,) it is a very costly and dangerous plan, and I do not think it will be adopted in mining anthracite.

Resumé.

From what has been already stated it will be seen that improvements are constantly being made in the methods of mining anthracite, but that these improvements are mainly confined to the details, and do not contemplate any change in the system of working.

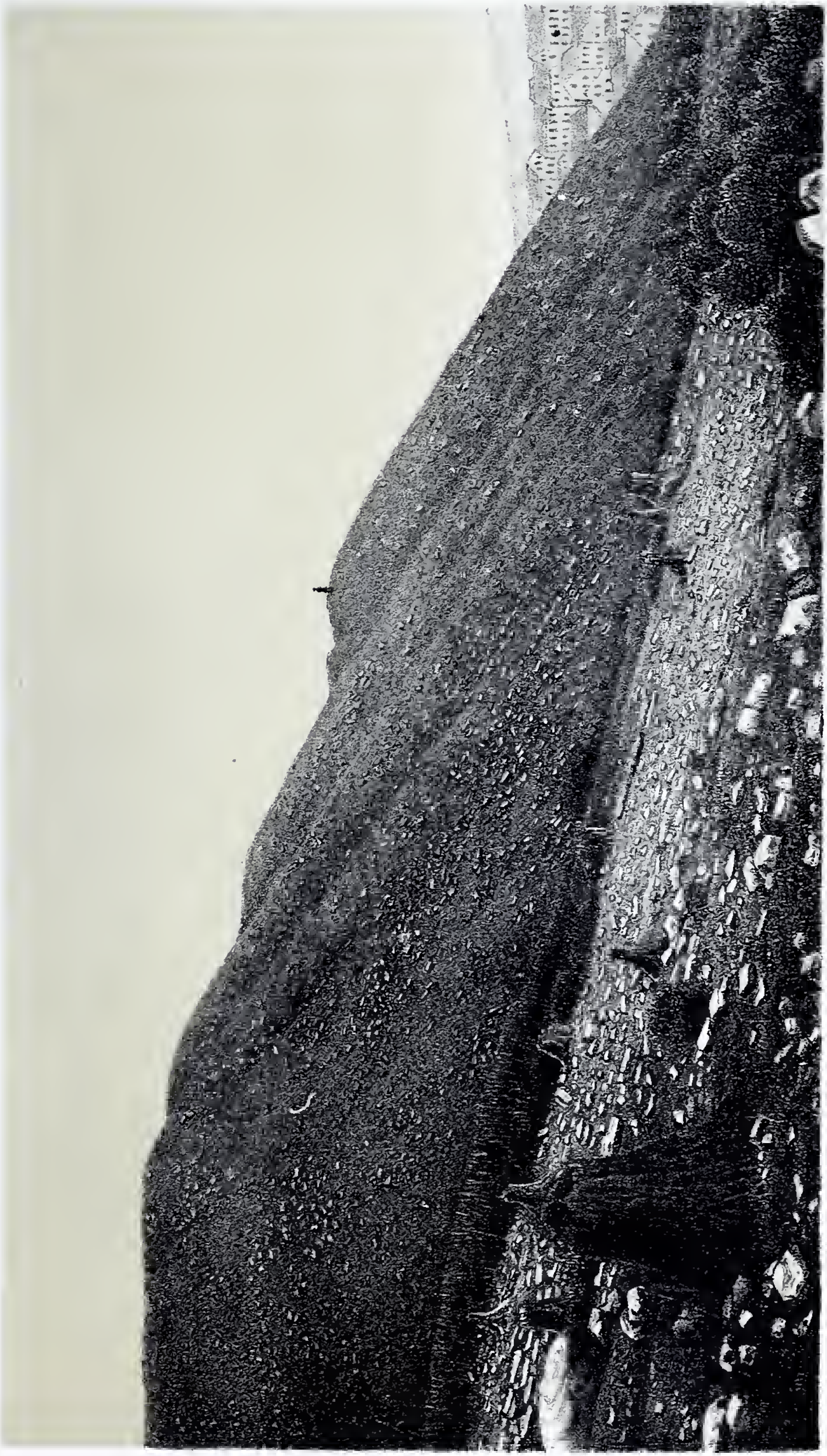
For mining beds of comparatively small size,—ten feet and less,—a very large percentage of the coal can be recovered by the ordinary plan of pillar and breast working, especially when the coal has a moderate inclination. Under such circumstances the waste from coal left in the mine in pillars and lost by roof-falls and crushes can probably be reduced to twenty *per cent.*

It is the mining of very thick beds, and beds of moderate thickness lying at high angles of dip, that present the greatest obstacles in the way of recovering a large percentage of the coal. Veith's boundary plan, the Rock-shute plan, the plan of driving to the limit before opening workings, and the plan of opening only every second or third breast while developing, all contemplate one and the same object, *viz.*—the localization of extensive roof-falls and crushes to areas nearly exhausted, and the prevention of gangway closure by a general "squeeze." The percentage of coal lost will be decreased by the general adoption of these plans (when successful) by, 1. The amount that would have been lost and ruined by crushes, and 2. By a certain variable quantity that can always be obtained when robbing can be carried to the utmost extreme, (the miner having no fear of inducing a general squeeze,) and when robbing is commenced as soon as the breasts are finished.

Without being able to give any general conclusions as to the reduction in waste that may be effected by the general adoption of one or all of these plans, I believe that in many localities the amount of coal won may be increased from ten to twenty-five *per cent.*

To effect any greater reduction in the percentage of waste I believe that the present system (pillar and breast) of mining must be replaced by some very different method.

Such a plan we already have in Col. Brown's panel sys-



CULM and ROCK HEAPS at SHENANDOAH

tem, but it is unfortunately not well adapted to beds dipping more than thirty-five degrees.

The long-wall system is not adapted to thick steep-pitching beds,—it is best adapted to the conditions from which the best results are now obtained in the ordinary pillar and breast workings.

Coal lost by becoming mixed with refuse and left in the mine (on moderate dips) is lost only because it would cost more than the coal is worth to separate it from the refuse. A discussion of this subject is unnecessary.

Much of the coal now lost by adhering to lumps of slate and other refuse rejected by the miner or at the breaker could be saved, but we cannot expect any operator to spend two or three dollars' worth of labor to save a ton of coal worth two dollars or less.

Waste made by blasting cannot well be decreased. At some collieries it might be somewhat diminished by limiting the amount of powder charged into a hole of a certain depth.

A certain percentage of fine coal must always be made by sliding the coal down shutes from the working place to the gangway. In loading the coal into cars and transporting it to the breaker, the amount of waste made varies with the softness or brittleness of the coal. In steep pitching beds this waste may be reduced to a minimum by drawing all the coal from the main shutes,—sending none down the manway,—or by keeping the manway (which then becomes only a shute) constantly full of coal.

As this latter plan cannot always be adopted with satisfactory results, the former method is often used.

We now come to a consideration of the means by which the breaker waste may be reduced.

It is evident at first sight that the coal should be moved in shutes, elevators, screens, etc., no further than is absolutely necessary to properly size and prepare the coal. This subject has already received attention in the preceding chapter, but I wish here to refer briefly to the use of very long screens. The percentage of waste always increases with the length of the screen, and while to insure proper sizing of the coal, long screens may be necessary with some varieties of coal, I am satisfied that good results might often be obtained with shorter screens. It also seems probable that the use of spiral screens might be feasible in some cases, and in others the larger sizes might be separated with less waste by double-jacked screens. However, as the amount of waste made by the screens is comparatively small, this is a matter of much less importance than the style of the rolls, their speed, etc.

It is a well demonstrated fact that the improved style of rolls with steel teeth, running at high speed, have effected a marked reduction in the percentage of breaker waste, and while we still look for further improvements in this direction, it is a matter of regret that the improvements already made have not been adopted at all anthracite collieries.

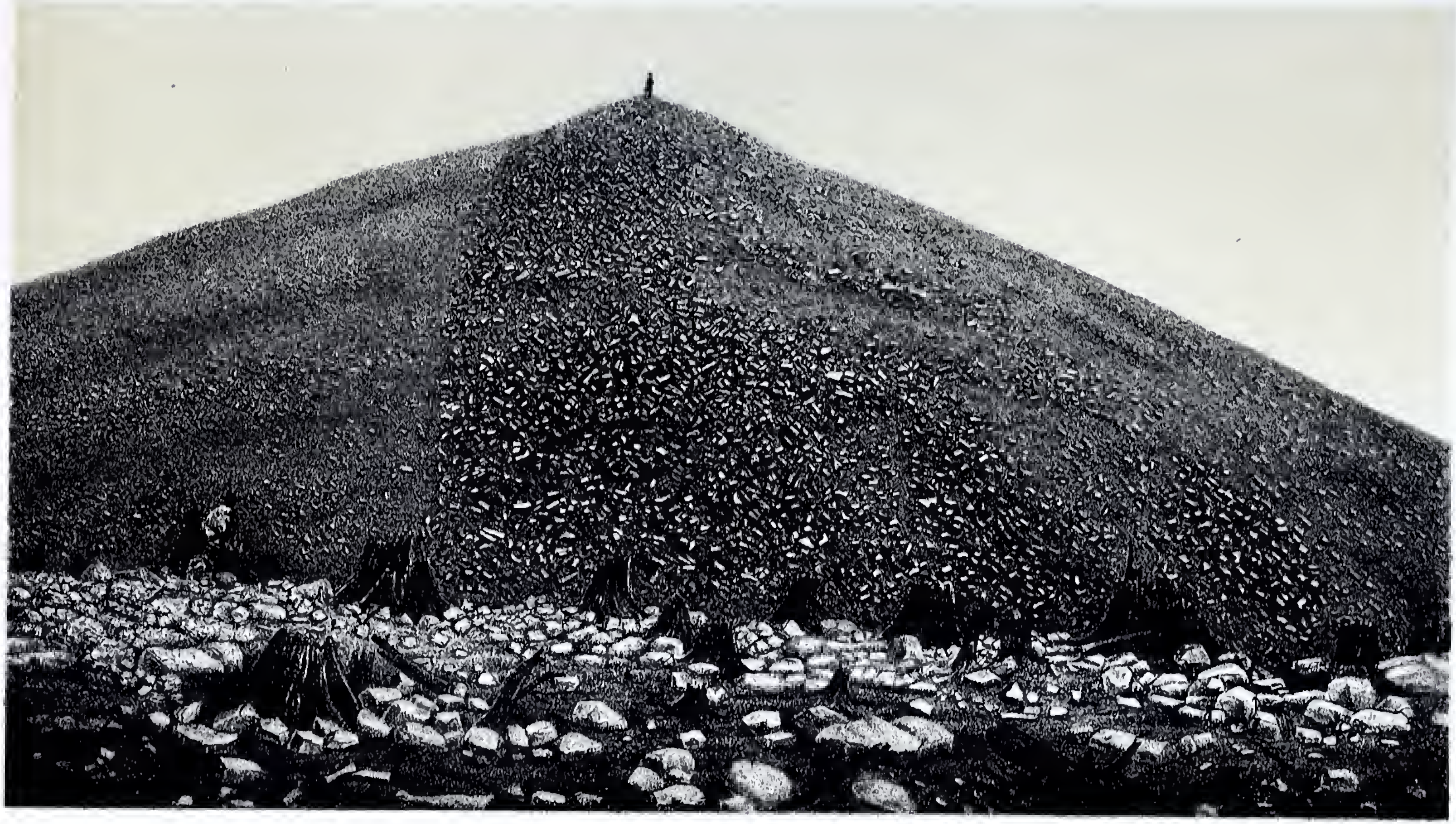
The waste made in loading the railway cars cannot well be avoided. I think the waste made by jarring during transportation is comparatively small,—probably much less than the waste made in loading and emptying the cars.

Utilization of Waste.

Of the total coal contained in the ground, I have estimated (see a preceding page) that

8 *per cent.* of waste is made by blasting and by handling the coal; and that
6.4 " is made by breaking and preparing the coal in the breaker, making
——
14.4 " of coal wasted by being broken into fine dust or dirt, (*culm.*) This fuel is waste, because it is in too fine a state to allow an ordinary draught to pass through it a sufficient quantity of air to support active combustion, and a strong (pressure) draught cannot be used, because it lifts the fire from the grate, and carries a large portion up the smoke stack.

Several plans for utilizing anthracite dust have been tried, with more or less success. With one exception they all



KOHINOOR COLLIERY CULM and ROCK DUMP.

resemble each other in two particulars, differing, however, in detail:

1. In them all, a cementing material is used to make the particles cohere; and

2. The compound is then compressed into lumps.

Clay, silicate of soda, (soluble glass,) and pitch have been used as cementing materials, but the necessity and difficulty of waterproofing the lumps made from clay, the failure of the silicate of soda to make a waterproof lump, and the objection common to both these plans of adding a non-combustible to the fuel, will probably prevent the future use of substances of this class. Pitch (coal tar) has given good results, and still better results are obtained by using coal tar and bituminous slack.

Square or rectangular lumps are objectionable because they pack together too closely, and the heat causes them to cohere into a mass too dense to permit free passage of sufficient air to keep up a hot fire. Round lumps are not easily handled. Oval or egg-shaped lumps, as made by the Loiseau process, have given the best results.

The works of the Loiseau Fuel Company, at Port Richmond, (Philadelphia,) have turned out a most excellent steam fuel; but their fuel cannot be compared to anthracite for domestic use. It can hardly be classed as an anthracite fuel; I should rather class it as a free-burning, non-coking, semi-anthracite or semi-bituminous coal.

It produces almost no clinker, and only a small percentage of ash, and has been shown to develop a calorific energy equal to ordinary anthracite coal.

It fires more quickly than anthracite; burns more quickly; can be banked up with slack (made by handling the fuel) like a soft coal fire; and is more easily raked than any coal I have ever seen. In brief, it is a very satisfactory fuel to fire with.

The cost of making this fuel will probably confine its manufacture for the present to the utilization of screenings made at seaboard shipping points, and in retail coal yards.

The quantity of material now lying in dirt-banks in the anthracite coal fields that might be utilized by this process has already been shown to be enormous.

Its utilization by this method will begin whenever the cost of manufacture is reduced to a point at which it can be made to compete with anthracite coal. It should bring about the same price as anthracite for steam and manufacturing purposes ; it must pay the same freight charges,—hence to make it at a profit, the total cost of manufacture must be reduced to or below the cost of mining and preparing ordinary coal for market.

The cost of gas tar, of bituminous slack, and the freight charges on same, will probably decide the future success or failure of this process in the anthracite regions.

Experiments on the consumption of dirt (and buckwheat) by locomotives with fire-boxes especially designed for this purpose, have shown that it is possible to maintain an excellent fire with good clean culm. A certain quantity of fine coal is now consumed in this way, but it represents only a very small percentage of the amount annually thrown away.

APPENDIX A.

MINE LAWS.

Inspection of Mines—Anthracite.

AN ACT providing for the health and safety of persons employed in anthracite coal mines.

Section 1. Be it enacted, etc., That the owner or agent of every anthracite coal mine or colliery shall make, or cause to be made, an accurate map or plan of the workings of such coal mine or colliery, on a scale of one hundred feet to the inch; and when there is more than one seam of coal worked in said coal mine or colliery, the map or plan shall exhibit the workings in each seam of coal, and shall state the general inclination of the strata, with any material deflection therein in said workings, and the boundary lines of the lands of said mines or colliery; a true copy of which map or plan, the said owner or agent shall deposit with the inspectors of coal mines and collieries for the district in which the coal mine or colliery is situated, within four months from the passage of this act, and one copy shall be kept at the office of each colliery; and the said owner or agent shall furnish to the inspector aforesaid, on the first day of January and July, in every year hereafter, a statement, or map or plan of the progress of the workings of such coal mine or colliery, during the year past, up to date, to enable the inspector to mark the same upon the map or plan of the coal mine or colliery, furnished and deposited with said inspector as herein provided for; and when any coal mine or colliery is worked out, preparatory to being abandoned, when any level or lift thereof is being finished, with a view and for the purpose of being abandoned, or when any of the pillars therein are to be removed, the owner or agent of such coal mine or colliery shall have the map or plan thereof furnished as hereinbefore provided, or such portions thereof as the case may require, carefully verified, and notice shall be given to the inspector of coal mines and collieries for the district, in writing, of the purpose to abandon or remove the pillars, as the case may be.

Section 2. That whenever the owner or agent of any coal mine or colliery shall neglect or refuse, or from any cause fail for the period of two months, to furnish to the inspector the map or plan, or the addition thereto provided for in the first section of this act, or if the inspector finds or has reason to believe, that any plan or map of any coal mine or colliery furnished him under the provisions of this act, is materially inaccurate or imperfect, he is hereby authorized to cause an accurate map or plan of the actual workings of such coal mine or colliery to be made, at the expense of the owner thereof, the cost of which shall be recoverable by law, as other debts are, from said owner.

Section 3. That four months from and after the passage of this act, it shall not be lawful for the owner or agent of any anthracite coal mine or colliery, worked by or through a shaft or slope, to employ any person in working within such coal mine or colliery, or to permit any person to be in such coal mine or colliery, for the purpose of working therein, unless there are in communication with every seam or stratum of coal worked in such coal mine or colliery, for the time being at work, at least two shafts, or slopes, or outlets, separated by natural strata, of not less than one hundred and fifty feet in breadth, by which shafts, slopes or outlets distinct means of ingress and egress are always available to the persons employed in the coal mine or colliery; but it shall not be necessary for the two shafts, slopes, or outlets to belong to the same coal mine or colliery, if the persons therein employed have ready and available means of ingress or egress by not less than two shafts, slopes or outlets, one or more of which may belong to another coal mine or colliery: *Provided,* That a second opening can be had through coal; but if a tunnel or shaft will be required for the additional opening, work upon the same shall commence immediately after the passage of this act, and continue until its final completion, with not less than three shifts in each twenty-four hours, and as many hands to be employed as can be put to work to advantage, the inspector to be the judge as to the least number of hands engaged per shift; this section shall not apply to opening a new coal mine or colliery, nor to any working for the purpose of making a communication between two or more shafts, slopes or outlets, so long as not more than twenty persons are employed at any one time in said new mine or working; and the term owner, used in this act, shall mean the immediate proprietor, lessee or occupier of a coal mine or colliery, or of any part thereof; and the term agent shall mean any person having, on behalf of the owner, the care or direction of a coal mine or colliery, or of any part thereof.

Section 4. The owner or agent of any coal mine or colliery, to which there is only one shaft, slope, or outlet, may petition the court of common pleas in and for the county in which such coal mine or colliery is situated, which said court is hereby empowered to act in the premises, setting forth that in consequence of intervening lands between the working of his coal mine or colliery and the most practicable point, or the only practicable point, as the case may be, at which to make or bring to the surface from the working of his mine, he is unable to make an additional shaft, slope, or outlet, in accordance with the requirements of this act; whereupon the court may make an order of reference, and appoint three disinterested persons, residents of the county, viewers, one or more of whom shall be practical mining engineers; all of whom, after being sworn to a faithful discharge of their duties, shall view and examine the premises, and determine as to whether the owner ought, or ought not, under the circumstances, to have the privilege of making an additional outlet through or upon any intervening lands. as the case may require, and report in writing to the next term of the court, which report shall be entered and filed of record; if the finding of the viewers, or any two of them, is in favor of the owner of such coal mine or colliery, he may make an additional shaft, slope, or outlet, under, through, or upon intervening lands, as may be determined upon and provided for by the award; if the finding of the viewers is against the owner, or if no award be made, by reason of any default or neglect on the part of the owner, he shall be bound to comply with the provisions of this act, in the same manner as if this section had not been enacted; in case the said owner or agent desires to, and claims that he ought to make an additional opening under, through, or upon any adjoining or intervening lands, to meet the requirements of this act, for the ingress or egress of the men employed in his or their coal mine or colliery, he or they shall make a statement of the facts in the petition, with a survey setting out the point of commencement and the point of termination of the proposed outlet, which he or they, their engineers, agents, and artists may enter upon said intervening land and survey, and mark as he or they shall find it proper to adopt for such additional outlet, doing no damage to the property explored; and the viewers shall state in their report what damage will be sustained by the owner or owners of the intervening lands by the opening, constructing, and using of the outlet, and if the report is not appealed from, it shall be liable to be confirmed or rejected by said court, as to right and justice shall appertain; and any further and all proceedings in relation thereto shall be in conformity with like proceedings as in the case of a lateral railroad across or under intervening lands, under the act in relation to lateral railroads, approved the fifth day of May, one thousand eight hundred and thirty-two, and the supplements thereto, so far as the provisions of the same are applicable hereto; and the notices to the owner of intervening lands of the intention to apply for the privilege of making an outlet and of the meeting of the viewers shall be given, and the costs of the case shall be paid, as provided in the said act of the fifth of May, eighteen hundred and thirty-two, and the supplements thereto.

Section 5. Any of the courts of law or equity of this commonwealth, having jurisdiction where the coal mine or colliery proceeded against is situated, upon application of the inspector of coal mines and collieries of the proper district, acting in behalf of the commonwealth, shall prohibit by injunction or otherwise, the working of any mine in which any person is employed in working, or is permitted to be for the purpose of working in contravention of the provisions of this act, and may award such costs in the matter of the injunction or other proceeding as the court may think just; but this section shall be without prejudice to any other remedy permitted by law for enforcing the provisions of this act.

Section 6. The owner, lessee, operator or agent of every coal mine or colliery, shall erect or provide, at or near the mouth or entrance to such mine, and maintain the same at all times when men are employed in such mine, a suitable building or buildings, supplied with soft water and properly lighted and warmed, for the use of the men employed in such mine, to wash and change their clothes when entering the mine and when returning therefrom.

Section 7. The owners or agent of every coal mine or colliery shall provide and establish for every such coal mine or colliery an adequate amount of ventilation of not less than fifty-five cubic feet per second of pure air, or thirty-three hundred feet per minute for every fifty men at work in such mine, and as much more as circumstances may require, which shall be circulated through to the face of each and every working place throughout the entire mine, to dilute and render harmless and expel therefrom the noxious, poisonous gases to such an extent that the entire mine shall be in a fit state for men to work therein, and be free from danger to the health and lives of the men by reason of said noxious and poisonous gases, and all workings shall be kept clear of standing gas; the ventilation may be produced by using blowing engines, forcing pumps, air pumps, forcing or suction fans of sufficient capacity and power, or other suitable appliances, as to produce and insure constantly an abundant supply of fresh air throughout the entire mine; but in no case shall a furnace be used in the mine where the coal breaker and schute buildings are built directly over and covering the top of the shaft for the purpose of producing a hot upcast of air; and there shall be an intake airway of not less than twenty square feet area, and the return airway shall not be less than twenty-five square feet.

Section 8. The better to secure the ventilation of every coal mine and colliery, and to provide for the health and safety of the men employed therein, otherwise and in every respect, the owner or agent, as the case may be, in charge of every coal mine or colliery, shall employ a competent and practical inside overseer, to be called mining boss, who shall keep a careful watch over the ventilating apparatus, over the airways, the traveling-ways, the pumps and sumps, the timbering, to see, as the miners advance in their excavations, that all loose coal, slate or rock overhead is carefully secured against falling, over the arrangements for signalling from the bottom to the top and from the top to the bottom of the shaft or slope, for the purpose of talking through, and all things connected with and appertaining to the safety of the men at work in the mine; he or his assistants shall examine carefully the working of all mines generating explosive gases every morning before the miners enter the coal mine or colliery, and shall ascertain that the mine is free from danger; and the workmen shall not enter the mine until such examination has been made and reported, and the cause of danger, if any exist, be removed; and he or his assistant shall also, every evening, when the workmen leave the mine or colliery, go over the mine and see that the doors of the passage ways are all properly closed and that all the air ways are free and unobstructed to the passage of air through them; and it shall be the duty of the mine boss to measure the ventilation at least once per week at the inlet and outlet, also at or near the face of all gangways; and all measurements to be reported to the inspector once per month.

Section 9. All and every of the safety-lamps used in coal mines or collieries shall be the property of the owner thereof, and shall be under the charge of a suitable person, under direction of the mining boss, who shall keep them clean and in good order; and the mining boss shall provide that all doors used in assisting, or in any way affecting ventilation of the mine, shall be so hung and adjusted as that they will close of their own accord, and cannot stand open; and the main air doors on the traveling roads shall be double, and an extra door shall be fixed, to be closed only in case of an accident to one of the others, and the side and top of such door shall be well built with stones and mortar, in mines in which the inspector shall deem it necessary and shall so order; and all main doors shall be provided with an attendant, whose constant duty shall be to guard them and prevent them being left open; and every mine having explosive gas, in each and every part of such mine or mines, shall be divided into two, four or more panels or districts, each ventilated by a separate split or current of air, and fifty persons shall be the greatest number that shall work in any one panel or district at the same time; and bore holes shall be kept twenty feet in advance of the face of each and every place, and if necessary, on both sides, when the same is driven towards or approaching an abandoned mine, or part of mine suspected to contain inflammable gases, or which is inundated with water.

Section 10. The owner or agent of every coal mine or colliery, opened and operated by shaft or slope, shall provide and maintain a metal tube from the top to the bottom of such slope or shaft, suitably calculated and adapted to the free passage of sound therein, through which conversation may be held by and between persons at the bottom and the top of the shaft or slope, and also the ordinary means of signalling from and to the top of the shaft from the bottom, and also provide an improved safety-catch and a sufficient cover overhead on every carriage used for lowering or hoisting persons; and they shall provide and arrange the flangs or horns of sufficient dimensions are attached to the sides of the drum of every machine that is used for lowering or hoisting persons in or out of every mine; an adequate break shall be attached to every drum or machine worked by steam or water power that is or will be used for lowering or raising into or out of any said mines, and the main link attached to the swivel of the wire, or any other rope, shall be made of the best quality of iron and tested by weights, or otherwise satisfactorily to the inspector, and bridle chains shall be attached to the main link from the cross pieces of the carriage, and no single link chain shall be used for lowering or raising persons into or out of any said mines; and no boy under twelve years of age shall work or enter any mine, and proof must be given of his age by certificate or otherwise, before he shall be employed; and no father or any other person shall conceal or misrepresent the age of any boy; the neglect refusal of any person or party to preform the duties provided for and required to be performed by sections six, seven, eight. nine and ten of this act, by the parties therein required to perform them, shall be taken and be deemed a misdemeanor by them, or either or any of them, and upon conviction thereof they or any or either of them shall be punished by imprisonment and fine of not exceeding five hundred dollars, or either, at the discretion of the court trying the same.

Section 11. No owner or agent of, or at any coal mine or colliery operated by shaft or slope, shall place in charge of any engine whereby the men are lowered into or hoisted out of the mine, any but experienced, competent, sober engineers; and every engineer so placed in charge of an engine shall constantly attend to the engine of which he has charge, and shall not allow any person, except such as may be deputed by the operator or agent, to touch or meddle with it or any part of its machinery; he shall work the engine slowly with great care when any person is ascending or descending the shaft or slope, and when any person is about to descend or ascend the shaft or slope, the men at the bottom, or top, as the case may be, must inform the engineer, by the metal tube, the signal, or otherwise. thereof; and no one shall interfere with, or in any manner intimidate the engineer in the discharge of his duties, nor ride upon a loaded wagon or cage, in any shaft or slope, and in no case shall more than ten men ride on any wagon or cage at one time in any of the said mines; and upon any person violating the provisions of this section he shall be held and deemed guilty of a misdemeanor, and upon conviction thereof he shall be punished by fine and imprisonment, at the discretion of the court trying the same.

Section 12. Whenever loss of life, or serious personal injury to any person shall occur, by reason of any explosion or other accident whatever, in or about any coal mine or colliery, it shall be the duty of the party having charge of such coal mine or colliery to give notice thereof forthwith, by mail or otherwise, to the inspector of coal mines and collieries for the district, and to the coroner of the county, if any person is killed thereby; and due notice shall be given, by the coroner, or any inquest to be held as the result of any such explosion or accident; and it shall be the duty of the said inspector to immediately repair to the scene of the accident and make such suggestions as may appear necessary to secure the safety of the men; and if the results of the explosion do not require an investigation by the coroner, he shall investigate into and ascertain the cause of the explosion or accident, and make a record thereof, which he shall preserve with other records of his office; and to eanble him to make the investigations, he shall have power upon such occasions to compel the attendance of persons to testify, and to administer oaths or affirmations thereto; the cost of which investigation shall be paid by the county in which the accident occured, in the same manner as costs of inquests held by the coroner or justices of peace are now paid; and the failure of the person in charge of the coal mine or colliery, to give notice to the inspector or coroner, as provided for in this section, shall subject him to a fine of not less than twenty-five, nor more than one hundred dollars, to be recovered as other fines are to the county treasury.

Section 13. All boilers used for generating steam in and about coal mines or collieries shall be kept in good order, and the owner or agent thereof shall have them examined and inspected by a competent boiler-maker, or other well qualified person, as often as once in six months, and oftener if needed, and the result of every such examination, under oath shall be certified in writing to the inspector for the district; and all machinery in and about the mines, and especially in the coal breakers, where boys work, shall be properly fenced off and the top of each shaft shall be securely fenced by vertical or flat gates, covering the area of said shaft, and the entrance of every abandoned slope, and air or other shaft, shall be securely fenced off.

Section 14. Upon the passage of this act, the governor of the commonwealth of Pennsylvania shall, upon the recommendation of a board of examiners, selected for that purpose, composed of three reputable coal miners in practice, and two reputable mining engineers, to be appointed by the judges of the court of common pleas of Luzerne county, all of whom shall be sworn to a faithful discharge of their duties, appoint three properly qualified persons to fill the offices of inspectors of coal mines and collieries for the counties of Luzerne and Carbon, whose commissions shall be for a term of five years or during good behavior, but they shall at all times be subject to removal from office for neglect of duty or malfeasance in the discharge of duty, as hereinafter provided for, and the persons so appointed shall have attained the age of thirty years, be citizens of Pennsylvania, and have a knowlege of the different systems of working coal mines, and have been intimately connected with the anthracite coal mines of Pennsylvania for a period of five years, and have had experience on the working and ventilation of coal mines where fire-damp and noxious gases are evolved; before entering upon their duties they shall take an oath or affirmation, before an officer qualified to administer the same, that they will perform the duties of the office with impartiality and fidelity, which oath or affirmation shall be filed in the office of the prothonotary of the county; and they shall provide themselves with the most approved modern instruments and chemical tests for carrying out the intentions of this act; the examiners provided for in this act shall be appointed by the judges of the court of common pleas for the county at the first term of the court in each year, to hold their places during the year, and vacancies shall be filled by the court as they occur; and the said examiners shall meet whenever candidates for the office of inspector are to be appointed, of which meetings public notice shall be given in at least two papers published in the county, at least two weeks before the meeting, the examiners shall agree in their recommendation of candidates to the governor, and they shall recommend only such as they find qualified for the office; the said examiners shall receive three dollars per day for every day they are actually engaged in the discharge of the duties of examiners under this act, to be paid to them by the county; one inspector shall be appointed for the district in the Wyoming coal field, Luzerne county, lying east of, and including Jenkins township, and one district shall be composed of that part of the Wyoming coal field lying west of Jenkins township and west of the Susquehanna river, and one other district shall be composed of that part of Luzerne county lying south of the Wyoming coal field, together with Carbon county.

Section 15. The term of office of the inspector of coal mines, appointed under an act for the better regulation and ventilation of mines, and for the protection of the lives of miners in the county of Schuylkill, approved April twelfth, one thousand eight hundred and sixty-nine, shall expire on the first day of June, Anno Domini one thousand eight hundred and seventy, and in his room three inspectors of mines for the counties of Schuylkill, Dauphin, Northumberland and Columbia, shall be appointed by examiners to be appointed by the court of common pleas of Schuylkill county, in the manner and form provided by the fourteenth section of this act; and the said examiners and inspectors, when so appointed, shall be subject to like regulations and duties, and entitled to like privileges, franchises and salaries as are in said section provided for the examiners and inspectors for the counties of Luzerne and Carbon; and the inspectors for the said counties of Schuylkill, Dauphin, Northumberland and Columbia, shall be assigned to duty in separate districts in said counties, which said districts shall be laid out and fixed by the examiners so as aforesaid to be appointed by the court of common pleas of the county of Schuylkill.

Section 16. It shall be the duty of the court of common pleas of the proper county, whenever a petition signed by not less than fifteen reputable coal operators or coal miners, or both, setting forth that any inspector of coal mines and collieries grossly neglects the duties, or that he is incompetent, or that he is guilty of malfeasance in office, to issue a citation in the name of the commonwealth, to the said inspector, to appear at not less than fifteen days' notice, on a day fixed, before said judges, when the said court shall proceed to inquire into and investigate the allegations of the petitioners; and if the court find that the said inspector is grossly neglectful of his duties, or that he is by any reason of causes that existed before the appointment, or that have arisen since his appointment, incompetent to perform the duties of said office, or that he is guilty of malfeasance in office, the court shall certify the same to the governor of the commonwealth, who shall declare the office of inspector of the district vacant, and proceed in compliance with the provisions of this act, to appoint a properly qualified person to fill the office; and the cost of the said investigation, before the court, shall be borne by the removed inspector; but if the allegations of the petitioners are not sustained by the final judgment of the court, the cost shall be paid by the petitioners.

Section 17. The salary of the said inspectors appointed for Luzerne and Carbon counties shall be three thousand dollars each; the maps and plans of mines and the records thereof, together with all papers relating thereto, shall be kept by the inspector, properly arranged and preserved in a convenient place in the district for which each inspector shall have been appointed.

Section 18. Each of the said inspectors of coal mines and collieries shall give his whole time and attention to the duties of the office and it shall be his duty to examine all the coal mines and collieries in his district, as often as his duties will permit him to do so, to see that every necessary precaution is taken to secure the safety of the workingmen, to see that the provisions of this act are observed and obeyed; and it shall be each inspector's duty to attend at every inquest held by coroner or coroners in his district, upon bodies of persons killed in or about coal mines or collieries.

Section 19. That any miner, workmen, or other person, who shall knowingly injure any safety lamp, water guage, barometer, air-course, brattice, or obstruct or throw open air-ways, or carry lighted pipes or matches in places that are worked by safety lamps, or hand or disturb any part of the machinery of the hoisting engine, or open a door and not have the same closed, whereby danger is caused in the mine, or enter any place of the mine against caution, or disobey any order given in carrying this act, or shall ride upon a loaded car or carriage in any shaft or slope, or on any plane in or around any of said mines, or do any other act whereby the lives or health of persons, or the security of the mines or machinery, is endangered; or any miner having charge of a working place in any coal mine or colliery, who shall neglet or refuse to keep the roof thereof properly propped and timbered, to prevent the falling of coal slate or rock, every such person shall be deemed guilty of a misdemeanor, and upon conviction shall be punished by imprisonment and fine at the discretion of the court.

Section 20. It shall be lawful for any inspector to enter, inspect, and examine any coal mine or colliery of his district, and the works and machinery belonging thereto, at all reasonable times, by day or night, but so as not to impede or obstruct the working of the coal mine or colliery, and to make inquiry into and touching the state and condition of such coal mine or colliery, works and machinery, and the ventilation and drainage of such coal mine or colliery, and the mode of lighting or using lights in the same, and into all matters and things connected with or relating to the health and safety of the persons employed in or about the same, and especially to make inquiry whether the provisions of this act are complied with in relation to such coal mine or colliery; and the owner or agent of such coal mine or colliery is hereby required to furnish the means necessary for such entry, inspection, examination, and inquiry, of which the said inspector shall make an entry in the record in his office, noting the time and material circumstances of the inspection.

Section 21. No person who shall act or practice as a land agent, or as a manager, viewer, or agent of any coal mine or colliery, or as a mining engineer, or be interested in operating any coal mine or colliery, shall at the same time act as inspector of coal mines and collieries under this act.

Section 22. It shall be the duty of each inspector to make an annual report of his proceedings to the governor of the commonwealth, at the close of every year, in which he shall fully enumerate all the accidents in and about the coal mines and collieries of his district, marking in tabular form those accidents producing death or serious injury to persons, and the state of the workings of said mines with regard to the safety of the workmen therein, and to the ventilation thereof, and the result of his labors generally shall be fully set forth.

Section 23. The salaries of each of the inspectors of coal mines and collieries, and the expenses of carrying into execution the provisions of this act shall be paid by the state treasurer, out of the treasury of the commonwealth, upon the warrant of the president judge of the court of common pleas of Luzerne county for the salaries of the inspector for Luzerne and Carbon counties, and upon the warrant of the president judge of the court of common pleas of Schuylkill county, for the inspectors for the counties of Schuylkill, Columbia, Northumberland and Dauphin; and all inspectors under this act shall reside in the district for which they are appointed.

Section 24. That for any injury to persons or property, occasioned by any violation of this act, or any willful failure to comply with its provisions by any owner, lessee or operator of any coal mine or opening, or on part of inspector to perform duty, a right of action shall accrue to the party injured, for any direct damages he may have sustained thereby; and in any case of loss of life, by reason of such violation or willful failure aforesaid, a right of action shall accrue to the widow and lineal heirs of the person whose life shall be lost, for like recovery of damages for the injury they shall have sustained.

Section 25. All laws of the commonwealth that are inconsistent with the provisions of this act are hereby repealed.

Approved the third day of March, Anno Domini 1870.

A SUPPLEMENT to an act entitled "An act providing for the health and safety of persons employed in coal mines," passed March 3, 1870.

Section 1. Be it enacted, etc., That inspectors of coal mines and collieries, provided for in the fourteenth section of the act, entitled "An act providing for the health and safety of persons employed in coal mines, approved the third day of March , Anno Domini one thousand eight hundred and seventy, shall and may be appointed by a majority of the board of examiners provided for in said act; should said board of examiners disagree in their selection of inspector, the judge of the court of common pleas of said counties have the power to dissolve the said board and appoint a new board of like qualifications, as aforesaid.

Section 2. All laws or parts of laws inconsistent with this act are hereby repealed.

Approved the 25th day of April, A. D. 1873.

A SUPPLEMENT to an act entitled "An act providing for the health and safety of persons employed in coal mines," passed March 3, 1870.

Section 1. Be it enacted, etc., That the first section of the said act be amended so as to read as follows, namely: That the owner or agent of every anthracite coal mine or colliery shall make or cause to be made, an accurate map or plan of the workings of such coal mine or colliery, on a scale of not more than one hundred feet to the inch, which map or plan shall exhibit the workings in each seam of coal, and shall state the general inclination of the strata, with any material defection therein in said workings, and shall truthfully and accurately show the boundary lines of the land of the said coal mines or colliery, and proximity of the workings to the lines of adjacent owners, a true copy of which map or plan the said owner or agent shall deposit with the inspectors of coal mines or collieries for the district in which the coal mine or colliery is situated, within four months from the passage of this act, and one copy shall be kept at the office of each colliery; and the said owner or agent shall furnish to the inspector aforesaid on the first day of January and July in every year hereafter, a statement or map or plan of the progress of the workings of such coal mine or colliery during the six months past up to date, to enable the inspector to mark the same upon the map or plan of the map, or plan of the coal mine or colliery furnished and deposited with said inspector as hereinbefore provided for, and when any coal mine or colliery is worked out preparatory to be abandoned, or when any local lift thereof is being finished with a view and for the purpose of being abandoned, the owner or agent of such coal mine or colliery shall have the map or plan thereof furnished as hereinbefore provided, or such portions thereof as the case may require, carefully verified, and notice shall be given to the inspector of coal mines and collieries for the district in writing of the purpose to abandon; and in case the said owner or agent shall neglect or refuse to furnish the maps or plans by this section required, or any of them, or shall knowingly and designedly cause such maps or plans when furnished to be incorrect or false, such owner or agent thus offending shall be guilty of a misdemeanor, and upon conviction shall be punished by a fine not exceeding five hundred dollars, or imprisonment not exceeding three months, at the discretion of the court.

APPROVED the 8th day of May, A. D. 1876.

AN ACT to amend section fourteenth of an act entitled "An act to provide for the health and safety of persons employed in coal mines," passed March 3, 1870.

Section 1. Be it enacted, &c., That section fourteenth of an act entitled "An act providing for the health and safety of persons employed in coal mines," be amended to read as follows: Upon the passage of this act the governor of the commonwealth of Pennsylvania shall, upon the recommendation of a board of examiners selected for that purpose, composed of three reputable coal miners in practice and two reputable mining engineers, to be appointed by the judges of the court of common pleas of Luzerne county, all of whom shall be sworn to a faithful discharge of their duties, appoint three properly qualified persons to fill the offices of inspectors of coal mines and collieries for the counties of Susquehanna, Wayne, Luzerne, and Carbon, whose commissions shall be for a term of five years or during good behavior; but they shall at all times be subject to removal from office for neglect of duty or malfeasance in the discharge of duty, as hereinafter provided for, and the persons so appointed shall have attained the age of thirty years, be citizens of Pennsylvania, and have a knowledge of the different systems of working coal mines, and have been intimately connected with the anthracite coal mines of Pennsylvania for a period of five years, and have had experience in the working and ventilation of coal mines where fire-damp and noxious gases are evolved. Before entering upon their duties they shall take an oath or affirmation before an officer qualified to administer the same, that they will perform the duties of the office with impartiality and fidelity, which oath or affirmation shall be filed in the office of the prothonotary of the county, and they shall provide themselves with the most approved modern instruments and chemical tests for carrying out the intentions of this act. The examiners provided for in this act shall be appointed by the judges of the court of common pleas for the county, at the first term of court in each year, to hold their places during the year, and vacancies be filled by the court as they occur; and the said examiners shall meet whenever candidates for the office of inspector are to be appointed, of which meetings public notice shall be given in at least two papers published in the county at least two weeks before the meeting; the examiners shall agree in their recommendation of candidates to the governor, and they shall recommend only such as they find qualified for the office; the said examiners shall receive three dollars per day for every day they are actually engaged in the discharge of the duties of examiners under this act, to be paid to them by the county. One inspector shall be appointed for the district in the Wyoming coal fields, which shall include Luzerne county lying east of including Jenkins township, and also so much of the counties of Wayne and Susquehanna, in which coal is mined or may hereafter be mined, and one district shall be composed of that part of the Wyoming coal fields lying west of Jenkins township and west of the Susquehanna river, and one other district shall be composed of that part of Luzerne county lying south of the Wyoming coal field, together with Carbon county.

APPROVED the 18th day of May, A. D. 1878.

AN ACT amendatory of an act entitled "An act providing for the health and safety of persons employed in coal mines, passed March 3, 1870.

Section 1. Be it enacted, &c., That section eleven of an act entitled "An act for the health and safety of persons employed in coal mines" approved the third day of March, one thousand eight hundred and seventy, which reads as follows: "No owner or agent of in or at any coal mine or colliery operated by shaft or slope, shall place charge of any engine whereby the men are lowered into or hoisted out of the mine any but experienced, competent, sober engineers; and every engineer so placed in charge of an engine, shall constantly attend to the engine of which he has charge, and shall not allow any person, except such as may be deputed by the operator or agent, to touch or meddle with it or any part of its machinery; he shall work his engine slowly and with great care when any person is ascending or descending the shaft or slope; and when any person is about to descend or ascend the shaft or slope the men at the bottom or top, as the case may be, must inform the engineer by the metal tube, the signal or otherwise, thereof; and no one shall interfere with or in any way intimidate the engineer in the discharge of his duties, or ride upon a loaded wagon or cage in any shaft or slope; and in no case shall more than ten men ride on any wagon or cage at any one time in any of said mines; and upon any person violating the provisions of this section, he shall be held and be deemed guilty of a misdemeanor, and upon conviction thereof he shall be punished by fine and imprisonment at the discretion of the court trying the same," be amended so as to read as follows: No owner or agent of or at any coal mine or colliery operated by shaft or slope, shall place in charge of any engine whereby the men are lowered into or hoisted out of the mine any but experienced, competent, sober engineers; and every engineer so placed in charge of an engine shall constantly attend to the engine of which he has charge, and he shall not allow any person except such as may be deputed by the operator or agent to touch or meddle with it or any part of its machinery; he shall work his engine slowly and with great care when any person is ascending or descending the shaft or slope, and when any person is about to descend or ascend the shaft or slope, the men at the bottom or top, as the case may be, must inform the engineer by the metal tube, the signal or otherwise, thereof; and no one shall interfere with or in any way intimidate the engineer in the discharge of his duties, nor ride upon a loaded wagon or cage in any shaft or slope; and whenever ten men shall have arrived at the bottom of any shaft or slope, they shall be furnished with an empty wagon or cage to ride up; and in no case shall more than ten men ride on any wagon or cage at one time in any of said mines; and upon any person violating the provisions of this section; he shall be held and be deemed guilty of a misdemeanor, and upon conviction thereof he shall be punished by fine and imprisonment at the discretion of the court trying the same.

APPROVED the 22d day of May, A. D. 1879.

AN ACT to provide for the selection of a site and erection of a State Hospital for injured persons, to be located in the counties of Schuylkill, Carbon, Dauphin, Northumberland, and Columbia, to be called the State Hospital for injured persons of the anthracite coal region, and for the management of the same, and making an appropriation therefor.

Section 1. Be it enacted, &c., That the governor shall appoint six commissioners to select a site and build a hospital for injured persons, to be located in the anthracite coal region, embracing the counties of Schuylkill, Carbon, Northumberland, Columbia, and Dauphin, two of said commissioners to be chosen from the county of Schuylkill, and one from each of the other counties embraced within the said anthracite coal region aforesaid, who shall serve without compensation.

Section 2. Said commissioners shall select, within four months of the date of their appointment, a tract of land within the said counties.

Section 3. Said tract of land so selected, shall be approved by the Governor in writing, and the deed for the same shall be taken in the name of the Commonwealth, in fee, for any land donated for the purpose aforesaid.

Section 4. The commissioners shall adopt such plan for said hospital as shall involve an expenditure, exclusive of the land, of not over sixty thousand dollars when completed: *Provided*, That the plan of the building shall be approved by the Board of Public Charities: *Provided further*, That the land before provided for shall be donated.

Section 5. The said commissioners shall have power to fix the salary of the superintendent of construction, and of such other per. sons as they may think necessary to employ in order to secure the proper and economical construction of the buildings: *Provided*, That the total expense of said buildings shall not exceed sixty thousand dollars

Section 6. To enable the commissioners to make necessary preparations for the erection of the buildings herein provided for, the sum of thirty thousand dollars is hereby appropriated annually for the years one thousand eight hundred and eighty and one thousand eight hundred and eighty-one, to be drawn from the treasury as may be required in the erection of buildings hereinbefore described, on warrants signed by the chairman of the commission, and countersigned by the president or general agent of the Board of Public Charities.

Section 7. Said commissioners shall proceed to erect said buildings and complete the same, at as early a period as possible compatible with the economical, substantial and skillful execution of the work, and shall make report to the Board of Public Charities of the amount of money expended by them and of the progress made in the erection of the buildings, semi-annually at least, and oftener if so required by said board.

Section 8. The said commissioners, upon the completion of said hospital, shall surrender their trust to a board of managers, to consist of nine members to be appointed by the Governor from the counties named in the first section of this act; said managers or trustees shall be a body politic or corporate, by the name and style of the trustees of the State Hospital for Injured Persons of the Anthracite Coal Region of Pennsylvania; they shall serve without compensation, and shall manage and direct the concerns of the institution, and make all necessary by-laws and regulations not inconsistent with the constitution and laws of the Commonwealth.

Section 9. That this hospital shall be specially devoted to the reception, care and treatment of injured persons, and that in the order of admission this class shall have precedence over paying patients.

Section 10. That the Governor, judges of the several courts of record of this Commonwealth, and members of the Legislature, shall be *ex-officio* visitors of the institution.

Approved June 11, 1879.

AN ACT to provide proper means of conveyance of persons, injured in or about the mines, to their homes.

Section 1. Be it enacted, &c., That from and after the passage of this act , every individual, firm or corporation, engaged in the mining of anthracite coal in this Commonwealth, shall keep at every colliery, worked by said individual, firm or corporation, except as hereafter provided, an ambulance or two stretchers, properly constructed, as the mine inspector of the district may deem the most suitable, for the purpose of conveying, to their homes or boarding houses, any person injured in or about the colliery or mine of such operator or operators, while engaged at his usual or temporary employment.

Section 2. If an ambulance, it shall be a closed vehicle with windows and shall be of sufficient size to convey at least two injured persons with two attendants, at the same time, and shall be provided with suitable springs, mattresses with roller beds, which may be removed at pleasure, into or from the vehicle, seats for the accommodation of attendants, and sufficient covering for the protection and comfort of the injured; and in all cases the injured person shall be conveyed to his home or boarding-house in said ambulance or stretcher, except as in cases hereinafter named.

Section 3. Such ambulance or stretcher shall be in charge of one of the superintendents of the colliery or collieries, and in his absence of some person convenient to the colliery and shall always be kept under cover and in readiness for use.

Section 4. In case the distance from the colliery, to the home or boarding-house of the injured person, is such that he may be quicker and more conveniently carried by railway, then such a mode of conveyance shall be permitted: *Provided always*, That such conveyance be under cover, and the comfort of the injured person be duly provided for.

Section 5. It shall be the duty of the mine inspectors of the several anthracite coal districts in the State to notify, immediately after the passage of this act, every individual, firm or corporation engaed in the mining of coal in their respective districts, of the requirements of this act, and, not later than six months after such notification, they shall personally visit every colliery in their respective districts, and in case they shall find that any operator or operators have neglected or refused to comply with the requirements of this act, the said inspector shall, or any other citizen may, forthwith institute proceedings against said operator or operators, before any alderman, magistrate or justice of the peace, in the county where such colliery or collieries are located.

Section 6. In case of the neglect or refusal of any individual, firm or corporation to comply with the requirements of this act, they shall be subject to a penalty of one hundred and fifty dollars, after hearing and conviction before any alderman or justice of the peace, on the report of said mine inspector, and in default of payment of the same shall be imprisoned for thirty days, unless defendant enter bail for his appearance at next term of quarter sessions, to be tried as for a misdemeanor, and upon conviction said court shall impose a fine not exceeding said one hundred and fifty dollars.

Section 7. This act shall not apply to individuals or companies employing less than twenty persons.

Section 8. When two or more collieries are situated not further than one mile apart, one ambulance shall be sufficient for such collieries: *Provided*, It is kept conveniently to each of them.

APPROVED May 10, 1881.

AN ACT making an appropriation for the hospital for injured persons of the anthracite coal region.

Section 1. Be it enacted, &c., That the sum of ten thousand dollars, in quarterly payments commencing June first, one thousand eight hundred and eighty-one, is hereby appropriated, and the state treasurer directed to pay the same, to the commissioners appointed to build a hospital for injured persons in the five counties of the anthracite coal region, approved June eleventh, one thousand eight hundred and seventy-nine, to enable the said commissioners to pay for the grading and excavating grounds, in addition to the cost of the building they the said commissioners were empowered to locate and build by the above mentioned act; also, for furnishing and maintaining the said hospital, the further sum of six thousand dollars for the year one thousand eight hundred and eighty-one, and six thousand dollars for the year one thousand eight hundred and eighty-two.

APPROVED June 29, 1881.

Inspection of Mines—Bituminous.

AN ACT providing the means for securing the health and safety of persons employed in the bituminous coal mines of Pennsylvania.

Section 1. Be it enacted, &c., That the owner or agent of every bituminous coal mine or colliery, shall make or cause to be made, within six months after the passage of this act, an accurate map or plan of the workings of such coal mine or colliery, on a scale not exceeding one hundred feet to the inch, and showing the bearings and distances, which shall be kept for use of the inspector, in the office at the mine of said coal mine or colliery; and said owner or agent shall cause, on or before the tenth of January in every year, a plan of the progress of the workings of such coal mine during the year past to be marked on the original map or plan of the said coal mine or colliery, and the inspector shall have the right at all times to have possession of any such map or plan at the mines to draw a copy therefrom for his own convenience: *Provided,* If the owner or agent of any coal mine shall neglect or refuse, or from any cause fail, for the period of two months after the time prescribed, to furnish the said map or plan as hereby required, or if the inspector shall find or have reason to believe that any map or plan of any coal mine furnished in pursuance of the provisions of this act is materially inaccurate or imperfect, he is hereby authorized to cause a correct map or plan of the actual workings of said coal mine to be made at the expense of the owner thereof, the cost of which shall be recoverable from said owner as other debts are recoverable by law: *Provided,* That if the map or plan which he claimed to be incorrect shall prove to be correct, then aforesaid expenses shall be paid by the said inspector and may be recovered from him in like manner.

Section 2. It shall not be lawful, after six months from the passage of this act, for the owner or agent of any bituminous coal mine to employ any person at work within said coal mine, or permit any person to be in said coal mine for the purpose of working therein, unless they are in communication with at least two openings separated by natural strata of not less than one hundred and fifty feet in breadth, if the mine be worked by shaft or slope, and of not less than twenty-four feet if the mine be worked by drift: *Provided*, If the mine be worked by drift two openings inclusive of air shaft shall only be required, if the air shaft can be used for ingress or egress in case of emergency; and that not more than twenty persons shall be employed in the mine at any one time after the expiration of the six months until the second opening shall be reached and made available; and in case of furnace ventilation being used before the second opening is reached, the furnace shall not be placed within forty feet of the bottom of the shaft, and shall be well secured from danger from fire by brick or stone walls of sufficient thickness and the mine while being driven for making and perfecting a second opening; the owner or agent shall provide and maintain a metal tube from the top to the bottom of the slope or shaft, suitably adapted to the free passage of sound, through which conversation may be held between persons at the bottom and at the top of the shaft or slope, also the ordinary means of signaling to and from the top and bottom of the shaft or slope, and an approved safety catch and sufficient cover overhead on every carriage used for lowering and hoisting persons; and the said owner or agent shall see that flanges or horns are attached to the sides of the drum of every machine that is used for lowering and hoisting persons in and out of the mine, and also that adequate brakes are attached thereto; the main link attached to the swivel of the wire rope shall be made of the best quality of iron, and shall be tested by weights or otherwise satisfactory to the inspector of mines of the district, and bridle chains shall be attached to the main link from the cross pieces of the carriage, and no single link chain shall be used for lowering or raising persons into or out of said mine, and not more than six persons shall be lowered or hoisted by the machinery at any one time; and only sober, competent and experienced engineers shall be employed.

Section 3. When a second opening is made one opening shall be set apart exclusively for purposes of ingress or egress, and shall not be clogged or obstructed with machinery, pumps, or currents of heated air or steam; if the opening is a shaft it shall be fitted with safe and convenient stairs, at an angle of not more than sixty degrees descent, and with landings at easy and convenient distances; all water coming from the surface or out of the stata in the shaft shall be conducted by rings or otherwise, to be prevented from falling down the shaft so as to wet persons who are ascending and descending the stairway of the shaft; if the opening is a slope, it shall be provided with safe and available traveling ways.

Section 4. The owner or agent of every bituminous coal mine, whether shaft, slope or drift, shall, within six months after the passage of this act, provide, and thereafter maintain for every such mine ample means of ventilation, affording one hundred cubic feet per minute for each and every person employed in said mine, which shall be circulated around the main headings and cross headings to an extent that will dilute, carry off and render harmless the noxious gases generated therein; and all mines generating fire-damp shall be kept free of standing gas, and every working place shall be carefully examined every morning with a safety lamp by a competent person before any workmen are allowed to enter.

Section 5. In order to better secure the proper ventilation of every coal mine, and promote the health and safety of the persons employed therein, the owner or agent shall employ a competent and practical inside overseer, to be called mining boss, who shall keep a careful watch over the ventilation apparatus, the air-ways, traveling-ways, pumps and pump timbers, and drainage, and shall see that as the miners advance their excavations, all loose coal, slate and rock overhead are carefully secured against falling in or upon the traveling-ways, and that sufficient timber is furnished of suitable lengths and sizes for the places where they are to be used, and placed in the working places of the miners; and it shall also be the duty of the mining boss to measure the air current at least once a week at the inlet and outlet and at or near the face of the heading, and keep a record of such measurements, and report the same to the inspector of his district once in every month; the safety lamps used for examining mines, or which may be used in working therein, shall be furnished by and be the property of the owner of said mines, and shall be in the charge of the agent of such mine; and in all mines generating explosive gases the doors used in assisting or directing the ventilation of the mine shall be so hung and adjusted that they will close themselves, or be supplied with springs or pulleys so they cannot be left standing open, and bore-holes shall be kept not less than twelve feet in advance of the face of every working place, and when necessary on the sides, if the same is driven towards and in dangerous proximity to an abandoned mine or part of a mine suspected of containing inflammable gases, or which is inundated with water.

Section 6. Any miners, workmen, or other persons, who shall intentionally injure any shaft, lamp, instrument, air-course, or brattice, or obstruct or throw open air-ways, or carry lighted pipes or matches into places that are worked by safety lamps, or handle or disturb any part of the machinery, or open a door and not close it again, or enter any place of the mine against caution, or disobey any order given in carrying out the provisions of this act, or do any other act whereby the lives or the health of persons, or the security of the mines or the machinery is endangered, shall be guilty of a misdemeanor, and may be punished in the manner provided in the sixteenth section of this act; all machinery about mines shall be properly fenced off, and the top of each shaft, and the entrance of every abandoned slope and air or other shaft shall be securely fenced off; and there shall be cut in the side of every hoisting shaft, at the bottom thereof, a traveling-way sufficiently high and wide to enable persons to pass the shaft in going from one side of the mine to the other without passing over or under the cage or other hoisting apparatus.

Section 7. If any person, firm, or corporation is, or shall hereafter be seized in his or their own right of coal lands, and it shall not be practicable to comply with the requirement of this act in regard to drainage and ventilation, by means of openings on his or their own lands, and the same can be done by means of openings on adjacent lands, he or they may apply by petition to the court of quarter sessions of the proper county, after ten days' notice to the owner or owners, their agent or attorney, setting forth the facts, under oath or affirmation, particularly describing the place or places where such opening or openings can be made, and that he or they cannot agree with the owner or owners of the land as to the amount to be paid for the privilege of making such opening or openings; hereupon the said court shall appoint three disinterested and competent citizens of the county to view the grounds designated, and lay out from the point or points mentioned in such petition a passage or passages for air and water, not more than sixteen feet in diameter, by the shortest and most convenient route to the coal of such person, firm, or corporation, preferring in all cases an opening through the coal strata where the same is practicable; the said viewers shall at the same time assess the damages to be paid by the petitioner or petitioners to the owner or owners of such land for the privilege of making said openings, which damage shall be fully paid before such opening is made; it shall be the duty of said viewers to give notice by at least three written or printed hand-bills, posted on the premises at least five days prior to the time of meeting to attend to the duties of their appointment, setting forth distinctly the time, place and object of their meeting, and also to give personal notice to the parties, their agents and attorneys, where it can be done, and shall, within thirty days after their appointment, make report of their proceedings to said court, stating the amount of damages awarded, accompanied by a map or plan of said openings; and if no appeal be taken to said court within ten days after notice to the opposite party in interest of the filing thereof, it shall be marked confirmed by the clerk, and the petitioner or petitioners may proceed to make said opening or openings; the pay of the viewers and other costs shall be the same as in road cases, and shall be paid by the petitioner or petitioners.

Section 8. As soon as practicable after the passage of this act, the persons exercising the office of president judge of each of the several courts of common pleas in the Fifth, Tenth, and Fourth judicial districts, shall appoint one reputable miner of known experience and in practice at the time, (in the Fifth district the president judge of the court of common pleas number one shall make said appointment,) and the governor shall appoint two mining engineers of like repute and experience and practice at the time, who shall constitute a board of five examiners, whose duty it shall be to inquire into the character and qualification of candidates for the office of inspector of mines under the provisions of this act; the examiners first appointed in pursuance of this section shall meet in the city of Pittsburgh on the fifteenth day of May next, and after being duly organized, having taken and subscribed before any officer authorized to administer the same the following oath, namely: "We the undersigned do solemnly swear (or affirm) that we will perform the duties of examiners of applicants for appointment as inspectors of bituminous coal mines to the best of our abilities, and that in recommending or rejecting said applicants we will be governed by the evidence of qualifications to fill the position under the law creating the same, and not by any consideration of political or other personal favors; that we will certify all whom we may find qualified according to the true intent and meaning of the act, and none others, to the best of our judgments," shall proceed to the examination of those who may represent themselves as candidates for said office; and they shall certify to the governor the names of all such applicants as they shall find competent to fill the office under the provisions of this act, which names, with the certificate and the oath of the examiners, shall be mailed to the secretary of the commonwealth to be filed in his office, and shall be valid when recommended by four of the examining board; the qualification of candidates for said office of inspector of mines to be inquired in and certified by said examiners, shall be as follows, namely: They shall be citizens of the United States, of temperate habits, of good repute as men of personal integrity, shall have attained the age of thirty years, and shall have had at least five years' experience in the workings of the bituminous coal mines of Pennsylvania, and upon the examination they shall give evidence of such theoretical as well as practical knowledge of the working of coal mines and noxious gases as will satisfy the examiners of their capability and fitness for the performance of the duties imposed upon inspectors of mines by the provisions of this act; the board of examiners shall also at their said meeting divide the bituminous coal counties of the State into three inspection districts, as nearly equal in regard to the labor to be performed as is possible, taking into consideration the number of mines and the extent of territory; at every subsequent calling of the board of examiners this division may be revised as experience may prove to be advisable; the board of examiners shall each receive five dollars per day and all necessary expenses, to be paid out of the state treasury upon the filing of the certificates of the examining board in the office of the secretary of the commonwealth, as hereinbefore provided; the governor shall, from the names so certified appoint one person to be inspector of mines for each district as fixed by the examiners in pursuance of the act, whose commission shall be for four years, to be computed from the fifteenth of May next; as often as vacancies occur by death, resignation or otherwise, in said offices of inspectors of mines, the governor shall fill the same by appointment for the unexpired term from the names on file in the office of the secretary of the commonwealth, until the number shall be exhausted, and whenever this shall occur the governor shall cause the aforesaid board of examiners to meet, who shall examine persons that may present themselves for the vacant office of inspector in the same manner as herein provided, and the board of examiners shall certify to the governor one person, to be commissioned by him for the office of inspector for the unexpired term; and any vacancies that may occur in the examining board shall be filled in the district where the vacancy occurred; and every four years the governor shall appoint two mining engineers as before, and shall notify the persons exercising the office of president judge of the courts of common pleas of the judicial districts of the state containing bituminous coal mines, selecting them in such order as to allow each district an equal share of such appointments, each to appoint one miner, and the five so appointed shall constitute a new board of examiners, whose duties, term of service and compensation, and vacancies that may happen, shall be the same as those provided for by this section; and from the names that may be certified by them, the governor shall appoint the inspectors provided for in this act; nothing in this act shall be construed to prevent the re-appointment of any inspector of bituminous mines; the inspectors of mines shall receive for their services an annual salary of two thousand dollars, to be paid quarterly by the state treasurer, and they shall reside in the district for which they shall be appointed; each inspector is hereby authorized to procure such instruments and chemical tests, and stationery, from time to time, as may be necessary to the proper discharge of his duties under this act, at the expense of the state, which shall be paid by the state treasurer upon accounts duly certified by him and audited by the proper department of the state. All instruments, plans, book memoranda, notes, et cetera, pertaining to the office, shall be the property of the state, and shall be delivered to their successors in office.

Section 9. Each inspector of bituminous coal mines shall, before entering upon the discharge of his duties, give bond in the sum of five thousand dollars, with sureties to be approved by the president judge of the district in which he resides, conditioned for the faithful discharge of his duty, and take an oath (or affirmation) to discharge his duties impartially and with fidelity to the best of his knowledge and ability.

Section 10. No person who shall act as a manager or agent of any coal mine or as a mining engineer, or to be interested in operating a coal mine, shall at the same time act as an inspector of mines under this act.

Section 11. For any injury to person or property occasioned by any violation of this act, or any willful failure to comply with its provisions by any owner, lessee or operator of any coal mine or opening, a right of action against the party at fault shall accrue to the party injured for the direct damage sustained thereby; and in any case of loss of life by season of such violation or willful failure, a right of action against the party at fault shall accrue to the widow and lineal heirs of the person whose life shall be lost, for like recovery of damages for the injury they shall have sustained.

Section 12. The inspectors of bituminous coal mines shall each devote the whole of his time to the duties of his office; it shall be his duty to examine the mines in his district as often as possible to see that all the provisions of this act are observed and strictly carried out; and he shall make a record of all examination of mines, showing the condition in which he finds them, the number of mines in his district, the number of persons employed in and about each mine, the extent to which the law is obeyed, the progress made in the improvement sought to be secured by the passage of this act, the number of accidents and deaths resulting from injuries received in the mines, and all other facts of public interest concerning the condition and progress of mining in his district, which record shall, on or before the first Monday of each month, together with all matters and things furnished him in accordance with the provisions of this act, be filed in the office of the Secretary of Internal Affairs, to be by him recorded and included in the annual report of his department; he shall also from the time of his commission make strict and careful inquiry and examination into the condition and drainage of the mines.

Section 13. That the inspectors may be enabled to perform the duties herein imposed upon them, they shall have the right at all times to enter any bituminous coal mine to make examination or obtain information; they shall notify the owners, lessees or agents immediately of the discovery of any violations of this act and of the penalty imposed thereby for such violation, and in case of such notice being disregarded for the space of ten days they shall institute a prosecution against the owner, owners, agent or lessee of the mine, under the provisions of section sixteen of this act; in any case however where, in the judgment of the inspector of either district, delay may jeopardize life or limb, he shall at once notify the inspectors of the other districts, whereupon they shall at once proceed to the mine or colliery where the danger exists and examine into the matter, and if after full investigation thereof they shall be agreed in the opinion that there is immediate danger, they shall apply in the name of the commonwealth to the court of common pleas of the county in which the mine may be located, for an injunction to suspend all work in and about such mine or colliery;

whereupon said court, if the cause appear to be sufficient, after hearing the parties and their evidence as in like cases, shall issue their writ to restrain the working of said mine or colliery until all cause of danger is removed; and the costs of said proceedings, including the charges of attorney prosecuting said application, shall be borne by the owner of the mine or colliery: *Provided*, That no fee exceeding the sum of twenty-five dollars shall be taxed in any one case for the attorney prosecuting such case: *Provided further*, That if said court shall find the cause not sufficient, then the case shall be dismissed, and the cost shall be borne by the inspector instituting the proceeding, or the county, in the discretion of the court.

Section 14. Whenever by reason of any explosion or other accident in any bituminous coal mine or the machinery connected therewith, loss of life or serious personal injury shall occur, it shall be the duty of the person having charge of such mine or colliery to give notice thereof forthwith to the inspector of the district, and if any person is killed thereby to the coroner of the county, who shall give due notice of the inquest to be held; it shall be the duty of the inspector upon being notified as herein provided, to immediately repair to the scene of the accident and make such suggestions as may appear necessary to secure the future safety of the men; and if the results of the explosion do not require an investigation by the coroner he shall proceed to investigate and ascertain the cause of the explosion or accident, and make a record thereof, which he shall file as provided for; and to enable him to make the investigation, he shall have power to compel the attendance of persons to testify, and to administer oaths or affirmations; the cost of such investigation shall be paid by the county in which the accident occurred, in the same manner as costs of inquests held by the coroners or justices of the peace are paid.

Section 15. The court of common pleas of any county in the proper district, upon a petition signed by not less than fifteen reputable citizens, not less than five of whom shall be miners, owners or lessees of mines, and with the affidavit of one or more of said petitioners attached, setting forth that any inspector of mines neglects his duties or is incompetent, or that he is guilty of a malfeasance in office, shall issue a citation in the name of the commonwealth to the said inspector to appear, on not less than fifteen days' notice, upon a day fixed before said court, at which time the court shall proceed to inquire into and investigate the allegations of the petitioners; if the court find that the said inspector is neglectful of his duties or is incompetent to perform the duties of his office, or that he is guilty of malfeasance in office, the court shall certify the same to the governor, who shall declare the office of said inspector vacant, and proceed in compliance with the provisions of this act to supply the vacancy; the costs of said investigation shall, if the charges are sustained, be imposed on the inspector; but if the charges are not sustained they shall be imposed upon the petitioners.

Section 16. The neglect or refusal to perform the duties required to be performed by any section of this act by the parties therein required to perform them, or the violation of any of the provisions or requirements hereof, shall be deemed a misdemeanor, and shall, upon conviction, be punished by fine of not less than two hundred dollars nor not exceeding five hundred dollars, at the discretion of the court, and all penalties recovered under this act shall be paid into the treasury of the state.

Section 17. The inspectors shall exercise a sound discretion in the enforcement of the provisions of this act; and should the operator or owner be dissatisfied with any decision at which the inspector may arrive, it shall and may be lawful for such operator or owner to apply by petition to the court of quarter sessions of the county wherein such mine is located, and said court shall thereupon appoint three reputable, competent, and disinterested persons, whose duty it shall be to forthwith examine such mines, and hear the proofs and allegations of the inspectors and operator or owner, and make such report under oath to court of the facts as they exist, together with their opinion thereon; and if said report sustains the decisions of the inspector, then the party making application to court shall pay the costs of such proceeding, and if the report is against such decision then the inspector shall pay the costs, unless the court order otherwise; the report of the said board shall become absolute unless exceptions thereto shall be filed within ten days after notice of the filing thereof to the owner, operator or inspector, and if exceptions are filed the court shall hear and determine the same, and the decision shall be final and conclusive.

Section 18. The provisions of this act shall not apply to any mine where ten men or a less number are employed, or to any mine which does not generate fire-damp, black-damp, or other dangerous or noxious gases.

Section 19. All laws or parts of laws inconsistent with any of the provisions of this act are hereby repealed.

Approved April 18, 1877.

AN ACT to amend an act, entitled "An act to provide the means for securing the health and safety of persons employed in the bituminous coal mines of Pennsylvania."

Section 1. Be it enacted, &c., That the last clause of the eighteenth section of the act of general assembly, entitled "An act to provide the means for securing the health and safety of persons employed in the bituminous coal mines of Pennsylvania," approved April 18, 1877, after the word "employed" in the second line of said section, which is as follows: "Or to any mine which does not generate fire-damp, black-damp, or other dangerous or noxious gases," is hereby repealed.

APPROVED May 25, 1878.

AN ACT to amend an act, entitled "An act to provide the means for securing the health and safety of persons employed in the bituminous coal mines of Pennsylvania," approved the eighteenth day of April, Anno Domini eighteen hundred and seventy-seven.

Section 1. Be it enacted, &c., That the eighth section of the act of general assembly, entitled "An act to provide the means for securing the health and safety of persons employed in the bituminous coal mines of Pennsylvania," approved the eighteenth day of April, Anno Domini eighteen hundred and seventy-seven, be and the same is hereby amended as follows: In the sentence "The board of examiners shall also at their said meeting divide the bituminous coal counties of the state into three inspection districts, as nearly equal in regard to the labor to be performed as is possible, taking into consideration the number of mines and the extent of territory," strike out and repeal the words "said" and "three" and insert in lieu thereof, respectfully, "next" and "four" so that said sentence will read: "The board of examiners shall also at their next meeting divide the bituminous coal counties of the state into four inspection districts, as nearly equal in regard to the labor to be performed as is possible, taking into consideration the number of mines and the extent of territory."

APPROVED June 3, 1881.

Corporation Stores.

AN ACT to provide for the incorporation and regulation of certain corporation.

Section 43. Every manufacturing, mining or quarrying company incorporated under the provisions of this act shall be confined exclusively to the purposes of its creation, as specified in its charter, and no such company shall manufacture or sell any commodity or articles of merchandise other than those therein specified. No such company shall engage in nor shall it permit any of its employés or officials to engage in the buying or selling upon the lands possessed by it of any wares, goods or commodities or merchandise, other than those specified in their charter or necessary for the manufacture of the same. No such company shall permit to be withheld or authorize or direct the withholding of wages due any of its operatives or employés by reason of the sale or furnishing of goods, wares or merchandise by any person to such operatives or employés, unless the same be withheld by reason of and in obedience to due process of law; but nothing herein contained shall prohibit any such company from supplying to its employés oil, powder, and other articles and implements necessary for or used in mining.

Approved April 29, 1874.

AN ACT fixing the standard weight for a bushel of bituminous coal in this Commonwealth.

Section 1. Be it enacted, &c., That from and after the passage of this act, the standard weight of bituminous coal in this Commonwealth shall be seventy-six pounds to the bushel, and two thousand pounds shall be one ton.

Section 2. If any person or persons engaged in the business of mining bituminous coal shall fix or establish, or shall attempt to fix or establish, any other number of pounds, by agreement or contract, to be a bushel of bituminous coal, than is provided for in the first section of this act, such person or persons shall be guilty of a misdemeanor, and upon conviction thereof, shall be sentenced to pay a fine, not less than five hundred and not exceeding one thousand dollars, and all penalties recovered under this act shall be paid into the treasury of the State.

Section 3. All acts or parts of acts inconsistent with this act are hereby repealed.

Approved May 18, 1878.

APPENDIX B.

A Glossary of Mining Terms.

In compiling the following glossary, an attempt has been made to include all of the most important technical terms in common use in the anthracite mining region and especially those common to both anthracite and bituminous mining.

To facilitate ready comparison I have also included the synonyms of many terms in use in metalliferous mining districts, and also some German and French terms.

Scientific terms not generally used by the practical miner, as well as those technical terms in common use among mechanics, architects, and civil engineers, have been excluded. For an explanation of such, reference may be had to any good technical dictionary.

At almost every colliery the miners have a peculiar series of terms (idioms) descriptive of the details of mining, driving, timbering, etc., but those used by miners at one locality are often unintelligible to those working in other districts, or even at times to those employed in an adjoining mine. As terms of this class are understood by the miners only, and often by a very limited number, no attempt has been made to include them in this glossary.

The abbreviations Eng., Fr., Ger., indicate that the term is in use in English, French, or German mining districts,—when inclosed in parentheses thus (Eng.,) that the term is common to both the American and foreign districts.

All metallurgical terms have been excluded as foreign to the objects of this glossary. For all such terms the reader is referred to Raymond's Glossary, in the Transaction American Institute of Mining Engineers, Vol. IX.

Glossary.

Adit, or Adit-level—A gangway driven in on a bed or vein from the surface, with just sufficient slope to insure drainage; generally from three inches to one foot fall in one hundred feet, or a tunnel driven in from the surface across the measures to open or drain a mine. Such a level is not, strictly speaking, a tunnel for a tunnel is an opening driven through from day to day, but in the United States the word tunnel is usually used in place of this more correct term.

After-damp—The mixture of gases remaining in a mine after an explosion of fire-damp, which may consist of carbonic acid gas, carbonic oxide, water vapor (quickly condensed) nitrogen, oxygen and in some cases free hydrogen, but usually consists principally of carbonic acid gas and nitrogen, and is therefore irrespirable.

Aérage, Fr.—Ventilation.

Air-heading, Eng.—Air-way.

Air-pipe or *Air-box*—Square boxes made of wooden boards in sections eight to sixteen feet long for the conveyance of air into headings and breasts; also iron pipes used for the conveyance of compressed air.

Air-stack—A ventilating chimney.

Air-way—Any passage used for the passage of air for ventilation.

Anemometer—An instrument used in measuring the velocity of the ventilating current of air.

Anticlinal—A flexure or fold in which the rocks on one side incline towards those on the opposite side of the fold, like the two legs of the letter A. The inclination on one side may be much greater than on the opposite side. An anticlinal is said to be *overturned* when the rocks on one side dip more than ninety degrees.

Anticlinal axis—The crest line of an anticlinal.

Anticlinal flexure; Anticlinal fold—An anticlinal.

Apex—The landing point at the top of a slope, the "knuckle;" also the top of an anticlinal fold.

Arenaceous—Sandy ; rocks are arenaceous when they contain a considerable percentage of sand.

Argillaceous—Clayey. An argillaceous rock is one that contains a considerable percentage of clay, or has some of the characteristics of clay.

Auger-stem—The iron rod or bar to which the bit is attached in rope-drilling.

Axis—See Anticlinal axis, and Synclinal axis. Often used synonomously with anticlinal ; thus the "Brady's Bend axis" for Brady's Bend anticlinal.

Backing deals, *Eng.*—Planks driven in behind the timbering in a shaft.

Balance-bob—A massive lever or rocking bob, weighted at one end to counterbalance the weight of the pump rods. In deep workings the weight of the rods is very great and balance-bobs are needed at intervals of from one hundred and fifty to three hundred feet.

Band, *Eng.*—Interstratified rock and coal.

Bank—A coal working opened by water-level drifts. Especially applied to bituminous mines. In English districts the area immediately surrounding the mouth of a shaft, the landing at the top.

Barney—*Ram*, *Mule*, *Truck*, *Donkey*—A small car used on inclined planes and slopes to push the mine car up the slope.

Barney-pit—A pit at the bottom of a slope or plane into which the barney runs to allow the mine car to run in over it, to the foot of the plane.

Barrier pillars—Large pillars of coal left at intervals to localize the damage resulting from a crush or squeeze.

Basin—A synclinal ; any depression of the strata not resulting from a fault ; the area containing coal.

Batt, *Eng.*—Shale ; hardened clay, but not fire-clay. Same as Bend and Bind.

Battery—Any structure built of timber or plank to keep the coal in the breast or to prevent it from sliding down a shute ; a platform on which the miners stand in thin steep-pitching beds of coal.

Bear ; to bear in—Underholing or undermining ; driving in at the top or at the side of a working.

Bearing—The strike, the course.

Bearing in—Usually applied to underholing. See *Bear.*

Bed—A regular member of a stratified series, deposited or formed after the underlying, and before the overlying rock.

Bed-rock—The solid rock underlying the soil, drift, or alluvial deposits.

Bench—A natural terrace marking the outcrop of any stratum ; a division of a coal seam separated from the remainder of the bed by a parting of slate, shale, iron pyrites, ("sulphur,") or other foreign matter.

Bind, Eng.—See *Batt.*

Bit—A drilling chisel. (See *auger-stem.*)

Black damp,—Choke-damp—Carbonic acid gas=CO_2 thus distinguished from White-damp or carbonic oxide=CO. See chapter on Mine Gases.

Blossom,—Smut, Outcrop, Tailing—The decomposed outcrop of a coal bed or mineral deposit.

Blower—A fan used for forcing air ; a strong discharge of gas from a fissure.

Blow-out, to blow out—A blast is said to blow out when it acts like a cannon, throwing out the tamping without bringing down the rock or coal.

Bob,—Pump-bob, balance-bob, rocking-bob—A triangular or four-sided frame of heavy timber or of iron, by which the horizontal motion communicated by the engine (connecting rod) is altered to the inclined or vertical motion of pump rods or of a man-engine.

Bone, Bony coal, Bony—Slaty or argillaceous coal, or carbonaceous shale occurring in coal seams.

Bonnet—A shield or covering over a cage to protect it and the miners from anything falling down the shaft.

Bord, Eng.—A breast.

Bord-and-Pillar, Eng.—See *Pillar and Breast.*

Bottom—The landing at the bottom of the shaft or slope, the lowest point of mining operations ; the floor, bottom rock, or stratum underlying a coal bed.

Bottom-lift—The lowest or deepest lift.

Brattice—A board or plank lining or partition in any mine passage to confine the air and force it into the working places. Its object is to keep the intake air from finding its way by a short route into the return airway.

Brattice-cloth—A heavy cloth or canvas, often covered with some water proofing material, for temporarily forcing the air into the face of a breast or heading; also used in place of doors on gangways; then known as "*sheets*."

Breast,—Chamber, Room—The excavation from which the coal is mined. (The word is sometimes applied to the working *face*.)

Breeding-fire—See *Gob-fire.*

Breaker—In anthracite mining, the structure in which the coal is broken, sized and cleaned for market. Known as also Coal Breaker.

Bridle-chains—Safety chains to support the cage if the draw-bar, the rope or link should break. When two chains are used on a slope (instead of attaching the rope by a single chain to the draw-bar) they are also called bridle-chains.

Broken Coal—In anthracite only—coal that is small enough to pass through a three and three eights to four inch (square) aperture, but that is too large to pass through a two and three-quarter or two and a half inch mesh. Smaller than steamboat, and larger than egg coal.

Brown Coal—Lignite. A fuel classed between peat and bituminous coal.

Bucket—The piston of a lifting pump.

Bucket-pump—A lifting pump. An iron or wooden receptacle for hoisting ore, or for raising rock in shaft sinking.

Buckwheat Coal—In anthracite only,—The smallest size of coal sent to market. Mostly made for use under boilers at the mines. Small enough to pass through a one-half or three-eighths inch mesh.

Buggy—A small wagon used for transporting coal from the working face to the gangway.

Bull-pump—A single acting pumping engine in which the steam cylinder is placed over the shaft or slope and the pump rods are attached directly to the piston rod. The steam enters below the piston and raises the pump rods; the water is pumped on the down stroke by the weight of the rods.

Bull-wheel—In rope drilling, the wheel used for raising the tools, for "spudding," etc.

Buntons—Timbers placed horizontally across a shaft or slope to carry the cage guides, pump rods, column pipe, brattice, etc.; also to strengthen the shaft timbering.

Butty, *Eng.*—Used to some extent in the U. S. A partner in a contract for driving or mining; comrade, crony.

Cable-drilling—*Rope drilling.*

Cage—A platform on which the mine cars are raised to the surface—see *Slope Cage.*

Cannel coal—See the Introductory Chapter, (Chapter I.)

Cap—*Cap-piece*—See *Collar.*

Cap—The pale bluish elongation of the flame of a candle or lamp caused by the presence of gas.

Carbonaceous—Coaly, containing carbon or coal.

Carboniferous—Containing or carrying coal, thus: Carboniferous rocks, the Carboniferous formation.

Car—*Mine-car*, *Wagon*, *Mine-wagon*—Any car used for the conveyance of coal along the gangways of a mine. In the anthracite regions they have a capacity of from seventy-five to one hundred and forty cubic feet.

Carriage—*Cage*—See *Slope Carriage.*

Cave, *Cave-in*—See *Crop-fall.*

Casing—Tubing inserted in a well to keep out the water or to protect the sides from collapsing.

CH_4—The chemical notation for Carburetted Hydrogen, or firedamp. See *Firedamp.*

Chain-pillar—A pillar left to protect the gangway and airway, and running parallel to these passages.

Chamber, *Room*, *Breast*—See *Breast.*

Charge—The amount of powder or other explosive used in one blast or shot.

Check Battery—A battery to close the lower part of a shute acting as a check to the flow of coal and as a stopping to keep the air in the breasts.

Cherry Coal—See Introductory Chapter, (Chapter I.)

Chestnut Coal—In anthracite only—Coal small enough to pass through a square mesh of one inch to one inch and an eighth, but too large to pass through a mesh of five eighths or half an inch. Known as No. 5 coal.

Chocks, Eng.—See *Nogs.*

Choke-damp—Carbonic acid gas, called also Black damp, chemical notation CO_2. See chapter on Mine Gases.

Chute—See Shute.

Clack—A pump valve.

Clack-piece—The casting forming the valve chamber.

Clack-door, (*Eng.*)—The opening into the valve chamber to facilitate repairs and renewals without unseating the pump or breaking the connections.

Clanny lamp—Safety lamp invented by Dr. Clanny.

Cleat, (*Eng.*)—The cleavage of a coal bed.

Cleavage—The property of splitting more readily in some directions than in others.

Clinometer—A small pocket instrument provided with a spirit-level and graduated arc for measuring the angle of dip. Made somewhat in the form of an ordinary foot-rule.

Clod—A layer of soft shale or slate forming a very bad roof to a coal bed. English, shale.

Clunch, Eng.—Under-clay, fire-clay.

Coal—See Chapter I of this Report.

Coal Measures—The Carboniferous formation.

Coal Breaker—See *Breaker.*

Collar—The horizontal timber resting upon two upright or inclined legs or props for the support of the roof in a gangway, airway, shute, slope or cross-heading. Called also the Cap or Cap-piece.

Colliery—This term is used to denote not only the mine, but includes also all the structures that make up the plant at the surface,—the mine and all its adjuncts.

Column-pipe—The large cast-iron (or wooden) pipes through which the water is conveyed from the mine pumps to the surface.

Conductors, Eng.—See *Guides.*

Conglomerate—The rock formation underlying the Coal Measures; a rock containing or consisting of pebbles, or of fragments of other rocks cemented together. English "Pudding Stone,"—Millstone Grit.

Cornish pump—A single-acting pump in which the motion is transmitted through a walking-beam, in other respects works like a Bull-pump.

Counter-gangway—A gangway driven obliquely across the workings to a higher level, or a gangway driven between two lifts and sending its coal down to the gangway below through a shute.

Counter-shute—A shute through which the coal from counter-gangway workings is lowered to the gangway below.

Country-rock—The rock surrounding or adjacent to an ore deposit.

Course—See *Strike.*

Coursing, or Coursing the air, is conducting it through the different portions of a mine by means of doors, stoppings, and brattices.

Cradle-dump—A rocking tipple for dumping cars. See *Dump.*

Creep, Eng.—A squeeze or crush forcing the pillars down into the floor or up into the roof, which often gives the miner the impression that the floor is rising.

Crevice—A fissure.

Crib, Crib-work—A structure composed of horizontal frames of timber laid upon one another, or a frame-work built like a log cabin.

Cribs—See *Nogs, Chocks, Packs.*

Cribbing—Timbering a shaft with crib-work timbering; commonly extends from the surface down to bed-rock.

Crop-fall—A caving in of the surface at the outcrop of the bed caused by mining operations. Applied also to falls occurring at points not on the outcrop of the bed. Synonomous with "Day-fall."

Crop—To come to the surface. An outcrop.

Cropping out—Coming to the surface; outcropping.

Cross-cut—In metalliferous mines, a level driven across the course of the vein. In coal mining used synonomously with Cross-heading.

Cross-heading—A passage driven for ventilation from the airway to the gangway, or from one breast through the pillar to the adjoining working.

Cross-hole—A cross-heading.

Crush—A general downward movement of the overlying measures caused by the pillars yielding to the weight of rock resting upon them; a squeeze.

Crushers, Crusher Rolls—See *Rolls.*

Culm—The English apply this name to anthracite coal. The fine coal, waste or "slack," coal dust and dirt now considered almost worthless.

Culm bank or *Culm dump*—Heaps of culm, now generally kept separate from the rock and slate dumps.

Dam—A timber bulkhead, or a masonry or brick stopping built to prevent the water in old workings from flooding other workings, or to confine the water in a mine flooded to drown out a mine fire.

Davy lamp—A safety lamp invented by Sir Humphrey Davy.

Day—A term used to signify the surface; thus, "driven to day," meaning to daylight, therefore to the surface.

Day-fall—See crop-fall.

Dead—The air of a mine is said to be dead or heavy where it contains carbonic acid gas, or when the ventilation is sluggish.

Dead-work—Work not in itself productive of enough coal or mineral to pay the cost of driving, or producing nothing at all. See *Narrow-work.*

Deep, Eng.—"To the deep," towards the lower portion of a mine, hence, the lower workings.

Derrick—The structure erected to sink a well or hole by the rope-drilling process.

Dig—To mine coal: applied to bituminous workings.

Diggings—Mining operations in bituminous coal (*rare.*)

Dip—The angle of inclination of a mineral bed or vein, measured from a horizontal line. The *direction of dip* is always at right angles to the *strike*.

Dividing Slate,—Parting—A stratum of slate separating two benches of coal bed.

Ditch, Drain—A gutter excavated in the floor of a gangway or airway to carry the water to the sump, or out to the surface.

Donkey engine—A small engine.

Door—Doors are placed in the passages of mines to prevent the ventilating current from taking a short cut to the upcast airway.

Door-tender—A boy whose duty it is to open and close a mine door before and after the passage of a train of mine cars.

Downcast—The passage or airway through which the ventilating current passes into a mine.

Draw—To "draw" the pillars; robbing out the pillars after the breasts are exhausted.

Draw-hole—An aperature in a battery through which the coal is drawn.

Drift—1. A water level gangway driven in on the bed from the surface; used synonomously with "entry" in bituminous mining; 2. Unstratified diluvial deposits, surface, soil, or other loose material.

Drill—Any tool used for boring or drilling holes in rock, coal, or mineral deposits.

Driving—Excavating horizontal passages,—in contradistinction to *sinking* or *raising*.

Drum—The revolving cylinder around which the winding rope is coiled.

Dumb-drift—An airway constructed to convey the ventilating current *around* the ventilating furnace to the upcast, instead of passing it directly through or over the fire.

Dump—1. A pile or heap of ore, coal, culm, slate, or rock; 2. The tipple by which the cars are dumped; 3. To unload a car by tipping it up.

Dumper—A car used on rock and culm banks, and constructed so that the body of the car may be revolved to dump the material in front, or on either side of the track.

Ear—The inlet or intake of a fan.

Egg coal—In anthracite only,—known also as No. 2 coal. Coal that is small enough to pass through a square mesh of two and a half or two and three quarters inches, but too large to pass through a mesh of two and a quarter inches.

Empties—Empty cars.

Empty track—Track used for empty cars.

Entry—A water level opening driven in on the bed. In bituminous mining applied to the main haulage road or gangway.

Face, or *Working face*—The place at which work is being done in a breast, gangway, airway, shute, or heading.

Fathom, *Eng.*—Six feet. In this county the foot and yard are used to the exclusion of this unit of measure.

Fault—See Chapter I.

Feeder—1. A spring of water encountered in sinking; 2 a small "*blower*" of gas.

Fire-board—A black-board on which the fire boss indicates every morning by chalk marks the amount of gas in different parts of the mine.

Fire-boss—A man whose duty it is to examine the workings for accumulations of explosive gas, etc.

Fire-clay—A stratum of clay frequently forming the floor of a coal bed.

Fire-damp—CH_4, light carburetted hydrogen. An inflammible gas, explosive when mixed with air in certain proportions. See the chapter on Mine Gases.

Floor—The rock underlying a coal seam; the "*hanging wall*" of any mineral deposit.

Fore-poling—Driving poles over the gangway timbers so that their ends project beyond the last set of timber, to protect the miner from roof falls; used also in quicksand or any loose material.

Free—Coal is said to be "free" when it is loose and easily mined, or when it will "run" without mining.

Gallows-frame—See *Head-frame.*

Gang—A set of miners, a "shift."

Gangway—A mine passage used for opening breasts, and for the haulage of coal.

Gas—Fire-damp.

Gas coal—Bituminous coal containing a large percentage of gas.

Goaf, *Eng.*—A space from which the coal has been mined; also the *gob*,—plural, goaves.

Gob—The refuse or waste left in the mine.

Gob-fire—Fire originating spontaneously from the heat of decomposing gob.

Grate coal—See *Broken coal.*

Guides—Vertical timbers fastened to the buntons to steady and guide the cage. English, "conductors." Wire rope guides or conductors, are common at English collieries.

Gunboat—A self-dumping box on wheels, used for raising coal in slopes; a monitor, a skip.

Hammer-and-Plate—A signaling apparatus. See chapter on Signaling apparatus.

Hanging Wall—The stratum lying on the upper side of a bed or vein,—used in metalliferous mining;—the roof.

Head-frame—A structure erected over a shaft to carry the sheaves and steady the cage. (Eng. Gallows-frame.)

Head-gear—That portion of the winding machinery attached to the head-frame, or the head-frame and its auxiliary machinery.

Head-house—When the head-frame is housed in, the structure is known by this name.

Heading—A cross-heading; a continuous passage for air or for use as a manway; the face where work is being done in driving any horizontal passage.

Head-piece—A cap, a collar.

Head-stocks—Gallows-frame, Head-frame.

Headways, *Eng.*—Cross-headings.

Heave, Eng.—A dislocation of the strata, throwing one portion of a bed or vein above or below, or to one side of the other portion ; a fault.

Hog-back—A sharp anticlinal axis of limited extent.

Holing—See Under-holing.

Holing-through—Driving a passage through to make connection with another part of the same workings, or with those in an adjacent mine.

Hopper—A coal pocket ; a funnel-shaped feeding trough.

Hood—See Bonnet.

Horse, or Horse-back—A portion of the roof or floor that bulges into the coal ; a mass of foreign material.

Horse-gin, or Horse-whim—A gearing for winding by horse-power.

H-piece—That portion of a *plunger-lift* pump containing the valves.

Incline—A slope, any inclined plane, whether above or beneath the surface ; usually applied to self-acting planes above ground, as in the bituminous coal fields.

Indicator—Any instrument or device for indicating the position of the cage in the shaft.

In place—A term used to denote the difference between loose matter not found at the outcrop, and that which is still "in place," and by which the outcrop is located.

Inside Slope—A slope on which coal is raised from a lower to a higher gangway.

Intake—A passage by which air enters a mine ; a downcast.

Jacket—The outside mesh of a double screen ; see chapter on the Preparation of Coal. Any casing or covering to prevent the radiation of heat. See Water-jacket.

Jars—In rope drilling—see Chapter II.

Jig—A machine for separating ores or minerals from worthless rock by means of their difference in specific gravity ; called also a "*jigger*" or "*washer.*"

Jigging—Separating ores or coal from refuse by jigs.

Jugglers—Timbers set obliquely against the rib in a breast to form a triangular passage to be used as a manway, airway or shute.

Jumper, Eng.—A drill operated by hand by "jumping" it up and down in the bore hole. It is sometimes weighted to give greater force to the blow, by making a swell near the middle or in the upper third of its length.

Keeps or Keps—Wings, catches, or rests, to hold the cage when it is brought to rest at the top, bottom or at an intermediate landing.

Kibble or Kibbal, Eng.—An iron bucket used in sinking or raising ore.

Lagging—Small round timber, slabs, or planks, driven in behind the legs and over the collar to prevent pieces of the roof from falling through.

Lander, Eng.—A man at the top who receives the mine car or bucket.

Landing—Any place where cars are taken off or put on a cage or slope.

Latches—A synonym of *switch*, applied to the split rail and hinged switches.

Legs—Props on which the collar rests in gangway and other timbering.

Level—A horizontal passage in a mine, or a drift driven in from the surface. Rarely applied to coal mining.

Lift—All of the workings and improvements opened at any one level at slope collieries. See Lifts.

Lifts—The number of gangways from which coal is raised at a slope colliery. The term originally referred to the number of pump lifts, but in the anthracite regions its significance has become broader. See Chapter on Methods of Working coal.

Lip Screen—A small screen or screen bars, placed at the draw hole of a coal pocket to take out the fine coal.

Loaded Track—Track used for loaded cars.

Long-wall—See Chapter on Methods of Working coal.

Loose-end—Mining so that the solid is open towards areas worked out on at least two sides.

Lump Coal—In anthracite only—All coal larger than Broken coal, or when Steamboat coal is made, lumps larger than this size.

Main-rod, Eng.—See Pump rod.

Main-way, Eng—A gangway.

Man-hole—A small and generally very short passage used only for the ingress and egress of the miners.

Man-engine—A machine for raising and lowering the miners. Operated like a set of pump rods, platforms being placed at regular distances for the miners to stand on. See description in this report.

Manway—A small passage used as a traveling way for the miner, and also often used as an airway or shute, or both.

Measures—Rocks, or a series of rocks, having some attribute in common; thus Coal Measures, the rocks containing coal, etc.

Mine—Any excavation made for the extraction of minerals or coal.

Mine road—Any mine track used for general haulage

Miner—This term is used in the anthracite fields to denote the workman who mines the coals. His *helpers* load the coal; they are also called *laborers.*

Mineral Charcoal—See Chapter I.

Monkey—Monkey shute, monkey drift, etc. The word "monkey" prefixed to a technical term means *small,* thus monkey shute, a small shute; monkey drift, a small drift—usually driven in for prospecting purposes.

Monitor,—Gunboat, Skip—See Gunboat.

Monoclinal—Applied to an area in which the rocks all dip in the same direction.

Mouth—The surface end of a shaft or drift.

Mounting-pipe—See *Column-pipe.*

Narrow-work—The driving and timbering of gangways, airways, cross-headings, etc., any *dead-work.*

Needle—An instrument placed in a bore-hole during the tamping of the charge, to leave on its withdrawal an opening through which the charge can be fired.

Nogs, Eng.—Logs of wood piled one on another to support the roof,—largely used in long-wall mining.

Nut coal—A contraction of the term *chestnut coal.*

Open cut—Any surface excavation.

Openings, an opening—Any excavation on a coal or ore bed; a mine.

Open-work—An open cut.

Operator—The individual or company actually working a colliery.

Outcrop—That portion of a vein, bed, or any stratum appearing at the surface, or occurring immediately beneath the soil or diluvial *débris*.

Outlet—A passage furnishing an outlet for air (upcast, out-take) for the miners, for water, or for the mineral mined.

Out-take—The passage by which the ventilating current is taken out of the mine; the *upcast*.

Output—The product of a mine sent to market, or the total product of a mine.

Overcast—A passage through which the ventilating current is conveyed over a gangway or airway.

Pack—A wall or pillar built of *gob* to support the roof; also used in the anthracite regions synononously with the English term *chocks* or *nogs*.

Panel, Panel working—See chapter on the Methods of Mining coal.

Parting—A layer of slate, bone, "sulphur" or other matter dividing two benches of a coal seam.

Pavement—The floor.

Pea Coal—In anthracite only,—coal small enough to pass through a mesh three quarters to half an inch square, but too large to pass through a three-eighths inch mesh. When Buckwheat coal is made, the size marketed as Pea is sometimes larger than the above; known also as No. 6 coal.

Picking Shute—A shute along which boys are stationed to pick the slate from the coal.

Picking Table—A flat or slightly inclined platform on which the coal is run to be picked free from slate.

Peter out—To "peter out" is to thin out, or gradually decrease in thickness.

Pillar-and-Breast, Pillar-and-Stall, Post-and-Stall, Bord-and-Pillar, etc., method of mining. See chapter on Methods of Mining coal.

Pillar—A mass of coal left to support the roof.

Pinch—To "peter out," to thin out; also, a *squeeze.*

Pipe—See air-pipe.

Pit, (*Eng.*—A shaft.) Any depression into which a cage, barney, or truck is lowered,—hence Cage-pit, Truck-pit, etc.

Pitch—See *Dip.*

Pit-frame—See *Head-frame.*

Plane—Usually applied to self-acting inclines, but any slope or incline on which coal is raised or lowered may be called a plane.

Plat, *or Plot*—A map of the surface and underground workings, or of either; to draw such a map from surveys.

Plunger—The cylindrical piston of a forcing pump.

Pocket—1, a hopper-shaped receptacle from which coal or ore is loaded into cars, canal boats, etc.; 2, a small body of ore.

Pole-tools—Drilling tools used in drilling in the old fashion with rods, now superseded by the rope-drilling method.

Post—Any upright timber; applied particularly to the timbers used for propping. See *Prop.*

Post and Stall—See *Pillar-and-Breast.*

Prop—A timber set upright or at right angles to the dip, to support the roof rock.

Proving-hole—1, a bore-hole drilled for prospecting purposes; 2, a small heading driven in to find a bed or vein lost by a dislocation of the strata, or to prove the quality of the ore in advance of regular workings.

Pulley—See *Sheave.*

Pulley-frame—See *Head-frame.*

Pump-bob—See *Bob.*

Pump-Slope—A slope used for pumping machinery.

Pump rods—Heavy timbers by which the motion of the engine is transmitted to the pump. In Cornish and Bull pumps, the weight of these rods makes the effective (pumping) stroke, the engine merely lifting the rods on the up-stroke. See *Balance-bob.*

Pump station—An enlargement made in a shaft, slope, or gangway, to receive the pump.

Red-ash coal—Coal that leaves a reddish ash.

Regulator—A frame with a sliding door to regulate the amount of air passing into any part of the workings

Rend-rock—A variety of dynamite.

Rests, *Keeps*, *Wings*—Supports on which the cage rests while the loaded car is being taken off and the empty one put on.

Rib—1. The wall or face forming the side of a working or passage ; the side of a pillar ; 2. A pillar.

Rider—A thin coal bed lying a few feet (or inches) above a workable seam.

Road—See *Mine road.*

Rob—To take out the pillars, or to reduce, by skipping, the size of the pillars left to support the roof.

Rock-shute—See *Slate shute*

Rolls—Heavy iron or steel cylinders, provided with teeth, for crushing or breaking large coal down to smaller sizes. They are made of several sizes : the largest are known as crushers, crusher rolls, or steamboat rolls , those of moderate size as prepared coal rolls, and the smaller rolls as monkey rolls.

Roof—The rock lying above a coal bed or ore vein : the *hanging wall.*

Room—See *Breast.*

Run—1. That portion of a counter-gangway (in flat workings) with a considerable grade ; 2. Coal is said to "run" when it breaks away from the solid without mining ; 3. "*By the run*," a method of mining in which the miners are paid by the lineal yard of breast driven, and not by the wagon ; it is employed in steep pitching workings.

Saddle—An anticlinal, a hog-back.

Safety-cage—A cage provided with an automatic safety-catch.

Safety-car—See *Barney.*

Safety-lamp—A lamp surrounded by wire gauze, to prevent the direct contact of the flame with explosive gases.

Sampson-post—In rope drilling, the post that supports the walking beam.

Sand-pump—A sludger; a cylinder provided with a stem (or other) valve, lowered into a drill hole to remove the pulverized rock.

Scale—The incrustation that forms in boilers using water containing mineral salts and acids.

Schute—See Shute.

Scraper—A tool used for cleaning bore-holes.

Screen—Any sieve, whether coarse or fine mesh, or bars, or perforated sheet metal, used for separating coal or ore into different grades according to size.

Seam—1. A bed of coal or mineral; 2. A fissure or joint, either empty or filled with foreign matter.

Shaft—A vertical opening sunk from the surface. In some mining districts this word is applied to openings not vertical.

Sheave—A wheel with a grooved circumference over which a rope is turned, either for the transmission of power or for winding or hauling.

Sheets—See *Brattice-cloth.*

Shelly—A name applied to coal that has been so crushed and fractured that it easily breaks up into small pieces.

Shift—1. The time for a miner's or laborer's work in one day; shifts vary from six to twelve hours. 2. The men working together in one shift.

Shot—A single hole, charged and fired.

Shoot--(See Shute.) To blast, or to fire a blast.

Show—A "show" of gas is a phrase denoting a quantity just sufficient to form a perceptible cap above the flame of a lamp or candle.

Shroud—A housing or jacket.

Shute—Any passage through which the coal descends by gravity. Applied somewhat improperly to the wagon way or entrance into flat workings. Written also chute shoot, and schute.

Silicious—Containing or having the characteristics of quartz (*silex.*)

Sinker-bar—In rope drilling—a heavy bar attached above the jars to give force *to the up-stroke.*

Skip or Skep—See *Gunboat.* A skip is usually made to run between guides, but in metalliferous mines the name is still applied when wheels and a track replace these guides.

Skip or "Skipping the Pillar"—To take a slice off the pillar before abandoning the workings; to rob.

Slack—Small coal or coal dirt. Applied usually to bituminous coal dirt.

Slant shutes—Shutes driven diagonally across to connect a breast manway with a manway shute.

Slate-picker—1. A man or boy who picks the slate and bony coal from the coal; 2. A segment of a cylindrical screen provided with narrow slits, through which the flat pieces of slate fall, but through which the coal (not being flat) cannot pass.

Slate shute—1. A shute for the passage of slate and bony coal to the pocket from which it is loaded into "dumpers"; 2. A shute driven through slate.

Slides—See *Guides.*

Slope—An inclined passage driven in the bed or vein, open at the surface; when not open at one end to the surface, known as an *inside slope.* See *Incline* and *Plane.*

Slope Cage—Slope Carriage—A truck on which the cars are raised at slopes on steep dips.

Soapstone—A term incorrectly applied by the miner to any soft unctious rock.

Splint Coal—See Chapter I.

Split—1. Any division or branch of the ventilating current; 2. The workings ventilated by that branch; 3. Any member of a coal bed split by thick partings into two or more seams; a *bench* separated by a considerable interval from the other benches of a coal bed.

Sprag—A short billet of wood used to lock the wheels of a mine-car in place of a brake. To insert sprags,—hence, to brake a car.

Sprag road—A mine road having such a sharp grade that sprags are needed to control the descent of the car,—hence Two, Three, or Four-Sprag-road.

Spring Latch—A spring or automatic switch.

Spud—A nail with a hole in the head driven into the mine timbers, or into a wooden plug fitted into the roof or floor to mark a surveying station.

Squeeze—A general settling down of the rocks overlying a mine or portion of a mine. See *Crush.*

Stall—See *Breast.*

Station—See *Pump Station* ; also *Bob-station*, etc.

Starter—The miner who ascends to the battery to start the coal to run.

Steamboat Coal—In anthracite only ; coal small enough to pass through bars set six to eight inches apart, but too large to pass through bars from three and one half to five inches. From this it will be seen that steamboat coal prepared at different collieries varies considerably in size. Comparatively few collieries make steamboat coal except to fill special contracts or orders.

Stoop-and-Room—See *Pillar-and-Breast.*

Stop—Any cleat or beam to check the descent of a cage, car, pump-rods, etc.

Stopping—A brattice, or more commonly, a masonry or brick wall built across old headings, shutes, airways, etc., to confine the ventilating current to certain passages, and also to lock up the gas in old workings, and in some cases to smother a mine fire.

Stove coal—In anthracite only ; two sizes of stove coal are made,—Large and Small. *Large Stove*, known as No. 3, *through* a $2\frac{1}{4}''$ to $2''$ mesh and *over* a $1\frac{7}{8}''$ to $1\frac{1}{2}''$ mesh ; *Small Stove*, known as No. 4, passes *through* a $1\frac{7}{8}''$ to $1\frac{3}{8}''$ mesh and over a $1\frac{1}{8}''$ to $1''$ mesh.

Stratum—Any bed or layer ; plural *Strata.*

Strike—The direction of a horizontal line drawn in any bed or vein ; its course.

Stripping—1. An open working ; 2. Removing the soil or *débris* on top of a bed preparatory to mining it by an open cut ; 3. The earth so removed.

Stump—A small pillar of coal left between the gangway or airway and the breasts to protect these passages ; any small pillar.

Stythe, Eng.—After-damp, choke-damp.

Sucker-rod—The pump-rod of an oil or artesian well.

Sulphur—1. Iron pyrites, bisulphide of iron; 2. Sulphurretted hydrogen, H_2S. Sometimes very improperly applied to *fire-damp*.

Sump, (from the German *Sumpf*,)—An excavation in the coal or rock made below the gangway to collect the mine water. The gangway ditches or drains empty into it, and the pump draws the water from it.

Swamp—A local depression in a coal bed in which the water collects. Applied particularly in bituminous mining.

Synclinal—An area in which the rocks incline away from each other like the two legs of the letter V. (See *Anticlinal.*) Used synonomously with *Basin.*

Synclinal axis—The line or course of a synclinal as determined by tracing a line through the lowest points along any stratum. (See Anticlinal axis.)

Tailing—The blossom, the outcrop or smut.

Telegraph—A trough-shaped shute for conveying coal or slate from the screens to the pockets.

Temper-screw—In rope drilling,—a screw for gradually lowering the clamped (upper) end of the rope as the hole is deepened.

Throw—1. A fault, a dislocation; 2. The amount of horizontal or vertical movement produced by a fault.

Tip, *Tipple*—The place where cars are tipped or dumped; the dump; a cradle-dump.

Top—The roof, the top coal, or top bench or benches; "*on top*," at the shaft or slope mouth, or on the surface.

Train or Trip—The number of cars taken by one team of mules, by a locomotive, or run at once on a slope, plane, or sprag road.

Trapper—A door-tender.

Traveling-way—A passage used by the miners for ingress and egress.

Transfer carriage, platform, or truck—Used to transfer mine cars.

Trip—See *Train.*

Trouble—A dislocation or fault; any irregularity in the bed.

Truck—Used synonomously with *Barney*.

Tunnel—A horizontal passage driven *across the measures* and open to day at both ends; applied also to such passages open to day at only one end, or not open to day at either end.

Turn—"To turn a breast," meaning to open or commence to open a breast.

Underclay—The clay, usually fire-clay, underlying a coal bed.

Underholing, *Undermining*—To mine out a portion of the bottom of a seam, by pick or powder, thus leaving the *top* unsupported and ready to be blown down by shots, broken down by wedges or mined with a pick or bar.

Unwater—To drain or pump the water from a mine.

Upcast—The opening or passage through which the air is taken out of a mine, the *Out-take*. The opposite of *In-take* and *Down-cast*. In-take and Out-take are generally applied to the *mine* airways; Down-cast and Up-cast to inclined or vertical airways open to the surface.

Vein—This term is often applied to stratified beds, but its use should be restricted to mineral deposits filling fissures or crevices in the rock.

Viewer, *Eng.*—A colliery manager or superintendent.

Wagon—A mine car. See *Car*, *Mine car*.

Wagon breast—A breast in which the mine cars are taken up to the working face.

Washer—A jig.

Water Guage—An instrument to measure the ventilating pressure; the term is also used to denote the ventilating pressure in inches.

Water jacket—See *Jacket*. A jacket filled with water to cool the cylinder.

Waste—Gob; also the fine coal made in mining and preparing coal for market; culm; coal dirt; dirt; also used to signify both the mine waste (or coal left in the mine in pillars, etc.,) and the breaker waste.

Water-level—Any passage driven with just sufficient grade to be used as a water-way; also a drift or gangway driven at the lowest level possible to carry the water directly out of the mine, through a tunnel or otherwise.

Well, Water-well—A sump, or a branch from the sump.

Whim—See *Gin* and *Horse-gin.*

White-ash Coal—Coal leaving a white-ash.

White-damp—CO. Carbonic oxide. A gas that may be present in the *after-damp*, or in the gases given off by a mine fire. Rarely met with in mines under other circumstances.

Win—To mine, to develop, to prepare for mining.

Winding—Hoisting coal or ore with a rope wound on a drum; used synonomously with *hoisting.*

Wings—See *Rests* and *Keeps.*

Winning, Eng.—A colliery, a new opening, a portion of a seam ready for actual mining; sometimes the portion mined.

Winze—An inside shaft, or very steep slope.

Work—To mine.

Worked-out—Exhausted.

Workings—Any species of development; usually restricted in meaning to apply to the breasts, etc., in contradistinction to the gangways and airways, but often used in a broader sense to mean all the *underground* developments.

Working face—See *Face.*

APPENDIX C.

Production and Distribution of Anthracite.

The following tables showing the production and distribution of anthracite coal are here reproduced from Miscellaneous Sheet No. III, lately published with the Panther Creek atlas.

The following notes also taken from the above-mentioned sheet will serve to explain the sub-division into districts:

"1. The Statistics prior to 1868 were collected and reported by Mr. P. W. Sheafer; since that date they have been compiled from official returns made regularly to Mr. J. H. Jones, Confidential Accountant of the Transporting Companies.

2. The Schuylkill Region includes the Western Middle Coal-field and Southern Coal-field east to Tamaqua.

3. The Lehigh Region includes the Southern Coal-field between Tamaqua and Mauch Creek and the Eastern Middle Coal-field.

4. The Wyoming Region embraces the Northern Coal-field.

5. No record has been made here of the production of soft Anthracite from the Loyalsock Coal-field.

6. The estimates given in these tables will be found to differ slightly from those reported by the Mine Inspectors."

The tables show a total tonnage for the different districts as follows:

	Tons.
Wyoming region,	181,484,879
Schuylkill region,	173,864,384
Lehigh region,	83,231,131
Total,	438,580,394

Anthracite Coal Tonnage of the Different Transportation Companies for the Years Named Below.

Compiled upon the Basis of Distribution established by the Anthracite Board of Control for 1878.

TRANSPORTING COMPANIES.	1870.	1871.	1872.	1873.	1874.	1875.
Philadelphia & Reading R. R. Company, ...	4,169,707	5,330,863	5,645,103	5,868,848	5,568,601	4,782,311
Lehigh Valley R. R. Company, ...	3,608,587	2,880,263	3,850,118	4,121,734	3,989,821	3,285,225
Central R. R. Company of New Jersey, ...	1,606,469	1,985,550	2,253,614	2,698,119	2,706,007	2,465,902
Delaware, Lackawanna & Western R. R. Company, .	2,117,612	1,730,242	2,520,330	2,952,941	2,353,539	2,833,670
Delaware & Hudson Canal Company, ...	2,318,073	1,955,737	2,882,479	2,732,267	2,290,791	2,843,229
Pennsylvania R. R. Company, ...	1,225,733	912,835	1,168,084	1,519,711	1,642,474	1,772,719
Pennsylvania Coal Company, ...	1,136,010	848,635	1,266,762	1,297,604	1,396,326	1,426,377
New York, Lake Erie and Western R. R, ...	. . .	55,596	83,288	36,728	197,562	303,039
Total, ...	16,182,191	15,699,721	19,669,778	21,227,952	20,145,121	19,712,472

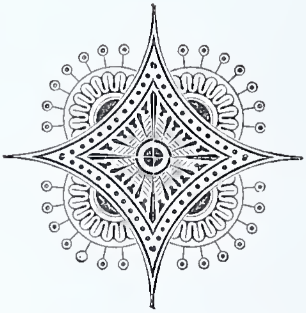
Anthracite Coal Tonnage of the Different Transportation Companies for the Years Named Below.

Compiled upon the Basis of Distribution established by the Anthracite Board of Control for 1878.

TRANSPORTING COMPANIES.	1876.	1877.	1878.	1879.	1880.	1881.
Philadelphia & Reading R. R. Company, ...	4,931,754	6,842,105	5,112,219	7,442,617	5,933,923	6,940,283
Lehigh Valley R. R. Company, ...	3,985,351	4,447,881	3,403,318	4,405,958	4,394,533	5,721,870
Central R. R. Company of New Jersey, ...	2,778,096	2,837,500	2,264,979	3,825,553	3,470,141	4,085,424
Delaware, Lackawanna & Western R. R. Company, .	1,998,654	2,089,523	2,180,673	3,867,405	3,550,348	4,388,970
Delaware & Hudson Canal Company, ...	1,809,190	1,787,470	2,046,235	3,014,117	2,674,705	3,211,496
Pennsylvania R. R. Company, ...	1,623,335	1,530,594	1,362,674	1,682,106	1,864,032	2,211,363
Pennsylvania Coal Company, ...	1,143,922	1,118,011	957,032	1,427,150	1,138,466	1,475,380
New York, Lake Erie & Western R. R., ...	230,709	175,095	278,132	477,783	411,094	465,230
Total, ...	18,501,011	20,828,179	17,605,262	26,142,689	23,437,242	28.500.016

Years.	Schuylkill Region.		Lehigh Region.		Wyoming Region.		Total Tons.
	Tonnage.	Per ct.	Tonnage.	Per ct.	Tonnage.	Per ct.	
1820, .	. . . . . .	. . .	365	. . .	. . . .	. . .	365
1821, .	. . . . .	. .	1,073	. . .	. . . .	. . .	1,073
1822, .	1,480	39.79	2,240	60.21	. . . .	. . .	3,720
1823, .	1,128	16.23	5,823	83.77	. . . .	. . .	6,951
1824, .	1,567	14.10	9,541	85.90	. . . .	. . .	11,108
1825, .	6,500	18.60	28,393	81.40	. . . .	. . .	34,893
1826, .	16,767	34.90	31,280	65.10	. . . .	. . .	48,047
1827, .	31,360	49.44	32,074	50.56	. . . .	. . .	63,434
1828, .	47,284	61.00	30,232	39.00	. . . .	. . .	77,516
1829, .	79,973	71.35	25,110	22.40	7,000	6.25	112,083
1830, .	89,984	51.50	41,750	23.90	43,000	24.60	174,734
1831, .	81,854	46.29	40,966	23.17	54,000	30.54	176,820
1832, .	209,271	57.61	70,000	19.27	84,000	23.12	363,271
1833, .	252,971	51.87	123,001	25.22	111,777	22.91	487,749
1834, .	226,692	60.19	106,244	28.21	43,700	11.60	376,636
1835, .	339,508	60.54	131,250	23.41	90,000	16.05	560,758
1836, .	432,045	63.16	148,211	21.66	103,861	15.18	684,117
1837, .	530,152	60.98	223,902	25.75	115,387	13.27	869,441
1838, .	446,875	60.49	213,615	28.92	78,207	10.59	738,697
1839, .	475,077	58.05	221,025	27.01	122,300	14.94	818,402
1840, .	490,596	56.75	225,313	26.07	148,470	17.18	864,379
1841, .	624,466	65.07	143,037	14.90	192,270	20.03	959,773
1842, .	583,273	52.62	272,540	24.59	252,599	22.79	1,108,412
1843, .	710,200	56.21	267,793	21.19	285,605	22.60	1,263,598
1844, .	887,937	54.45	377,002	23.12	365,911	22.43	1,630,850
1845, .	1,131,724	56.22	429,453	21.33	451,836	22.45	2,013,013
1846, .	1,308,500	55.82	517,116	22.07	518,389	22.11	2,344,005
1847, .	1,665,735	57.79	633,507	21.98	583,067	20.23	2,882,309
1848, .	1,733,721	56.12	670,321	21.70	685,196	22.18	3,089,238
1849, .	1,728,500	53.30	781,556	24.10	732,910	22.60	3,242,966
1850, .	1,840,620	54.80	690,456	20.56	827,823	24.64	3,358,899
1851, .	2,328,525	52.34	964,224	21.68	1,156,167	25.98	4,448,916
1852, .	2,636,835	52.81	1,072,136	21.47	1,284,500	25.72	4,993,471
1853, .	2,665,110	51.30	1,054,309	20.29	1,475,732	28.41	5,195,151
1854, .	3,191,670	53.14	1,207,186	20.13	1,603,478	26.73	6,002,334
1855, .	3,552,943	53.77	1,284,113	19.43	1,771,511	26.80	6,608,567
1856, .	3,603,029	52.91	1,351,970	19.52	1,972,581	28.47	6,927,580
1857, .	3,373,797	50.77	1,318,541	19.84	1,952,603	29.39	6,644,941
1858, .	3,273,245	47.86	1,380,030	20.18	2,186,094	31.96	6,839,369
1859, .	3,448,708	44.16	1,628,311	20.86	2,731,236	34.98	7,808,255
1860, .	3,749,632	44.04	1,821,674	21.40	2,941,817	34.56	8,513,123
1861, .	3,160,747	39.74	1,738,377	21.85	3,055,140	38.41	7,954,264
1862, .	3,372,583	42.86	1,351,054	17.17	3,145,770	39.97	7,869,407
1863, .	3,911,683	40.90	1,894,713	19.80	3,759,610	39.30	9,566,006
1864, .	4,161,970	40.89	2,054,669	20.19	3,960,836	38.92	10,177,475

Years.	SCHUYLKILL REGION.		LEHIGH REGION.		WYOMING REGION.		TOTAL.
	Tonnage.	Per ct.	Tonnage.	Per ct.	Tonnage.	Per ct.	Tons.
1865, .	4,356,959	45.14	2,040,913	21.14	3,254,519	33.72	9,652,391
1866, .	5,787,902	45.56	2,179,364	17.15	4,736,616	37.29	12,703,882
1867, .	5,161,671	39.74	2,502,054	19.27	5,325,000	40.99	12,988,725
1868, .	5,330,737	38.62	2,502,582	18.13	5,968,146	43.25	13,801,465
1869, .	5,775,138	41.66	1,949,673	14.06	6,141,369	44.28	13,866,180
1870, .	4,968,157	30.70	3,239,374	20.02	7,974,660	49.28	16,182,191
1871, .	6,552,772	41.74	2,235,707	14.24	6,911,242	44.02	15,699,721
1872, .	6,694,890	34.03	3,873,339	19 70	9,101,549	46.27	19,669,778
1873, .	7,212,601	33.97	3,705,596	17.46	10,309,755	48.57	21,227,952
1874, .	6,866,877	34.09	3,773,836	18.73	9,504,408	47.18	20,145,121
1875, .	6,281,712	31.87	2,834,605	14.38	10,596,155	53.75	19,712,472
1876, .	6,221,934	33.63	3,854,919	20.84	8,424,158	45.53	18,501,011
1877, .	8,195,042	39.35	4,332,760	20.80	8,300,377	39.85	20,828,179
1878, .	6,282,226	35.68	3,237,449	18.40	8,085,587	45.92	17,605,262
1879, .	8,960,829	34.28	4,595,567	17.58	12,586,293	48.14	26,142,689
1880, .	7,554,742	32.23	4,463,221	19.05	11,419,279	48.72	23,437,242
1881, .	9,253,958	32.46	5,294,676	18.58	13,951,383	48.96	28,500,017



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